DESIGN OF SYSTEMATIC SUPPORT SYSTEM FOR DEVELOPMENT AND DEPILLARING IN UNDERGROUND COAL MINES

A THESIS SUBMITTED IN PARTIAL FULFILLMENT

OF THE REQUIREMENTS FOR THE DEGREE OF

BACHELOR OF TECHNOLOGY

IN

MINING ENGINEERING

BY

VIKRANT DEV SINGH

109MN0016

Department of Mining Engineering

National Institute of Technology

Rourkela

2013

DESIGN OF SYSTEMATIC SUPPORT SYSTEM FOR UNDERGROUND DEVELOPMENT AND DEPILLARING IN COAL MINES

A THESIS SUBMITTED IN PARTIAL FULFILLMENT

OF THE REQUIREMENTS FOR THE DEGREE OF

BACHELOR OF TECHNOLOGY

IN

MINING ENGINEERING

BY

VIKRANT DEV SINGH

109MN0016

Under the Guidance of

Prof. S. Jayanthu & Prof. D. P. Tripathy

Department of Mining Engineering

National Institute of Technology

Rourkela

2013

National Institute of Technology

Rourkela

CERTIFICATE

This is to certify that the thesis entitled,-"***Design of Systematic Support System for Development and Depillaring in Underground Coal Mines***" submitted by **Mr. Vikrant Dev Singh**, **109MN0016**, in partial fulfillment of the requirement for the award of Bachelor of Technology Degree in Mining Engineering at the National Institute of Technology, Rourkela (Deemed University) is an authentic work carried out by him under our supervision and guidance.

To the best of my knowledge, the matter embodied in the thesis has not been submitted to any University/Institute for the award of any Degree or Diploma.

Prof. D. P. Tripathy

Department Of Mining Engineering

National Institute of Technology

Rourkela – 769008

Prof. S. Jayanthu

Department Of Mining Engineering

National Institute of Technology

Rourkela – 769008

Date:

Date:

ACKNOWLEDGEMENT

My heart pulsates with the thrill for tendering gratitude to those persons who helped me in construction of the project. Unfortunately, the list of expressions of thank no matter how extensive is always incomplete and insufficient.

First and foremost, I express my sincere appreciation and gratitude to Prof. S. Jayanthu and Prof. D. P. Tripathy, Professors of Department for allowing me to carry on the present topic "Design of systematic support system for development and depillaring in underground coal mines" and later on for their inspiring guidance, constructive criticism and valuable suggestions throughout this project work. I am very much thankful to them for their able guidance and pain taking effort in improving my understanding of this project.

I would also like to extend our sincere thanks to the Mr. K. Narayana, Mines Manager of RK-6 Inc, SCCL and other officials, who helped me during my sample and data collection in their respective regions.

An assemblage of this nature could never have been attempted without reference to and inspiration from the works of others whose details are mentioned in reference section. I acknowledge my indebtedness to all of them.

Last but not the least; I would like to thank all my friends who have patiently extended all sorts of helps for accomplishing this project.

Date:

Vikrant Dev Singh

109MN0016

Department of Mining Engineering

National Institute of Technology

Rourkela-769008

CONTENTS

ABSTRACT

Bord and pillar method is the most widely practiced underground mining method in India. Nevertheless about 61% of underground coal mining accidents are due to roof and side fall of bord and pillar. Design of systematic support is essential to avoid strata control problem and to provide safe working condition. Three distinct methodologies; empirical approach, numerical modeling and field monitoring were followed and compared in the project to provide a comprehensive design of systematic support. A case study of 1AS2 panel of RK-6 Incline, SCCL is chosen for the design of systematic support.

The empirical design of support is developed using RMR and Q-system and validated with numerical modeling and field monitoring. The systematic support developed by RMR with factor of safety greater than 2 for the gallery was 1.5 m spacing of 1.8 m full column grouted bolts with spacing of 1.4 m between rows. Junction support was 33% extra full column grouted bolts. Systematic support designed for slices and goaf edges with Q-System was skin to skin chocks with corner props and breaker line bolt with 1 m spacing.

Maximum convergence measured with telescopic convergence rod in the field at the station 6F- 57LS was 48 mm. Maximum deformations observed by numerical modeling in the gallery was 58 mm. The numerical model was almost validated with the field monitoring data with 17% approximation, thus the numerical model can be used for prediction of strata behavior of future working.

LIST OF TABLES

LIST OF FIGURES

CHAPTER 1

INTRODUCTION

INTRODUCTION

Underground coal mining in India is predominantly carried by bord and pillar method. It contributes over 90% of the underground coal working today and is expected to prolong in future. Bord and pillar method is popular for flat deposits with thin seam, but its safety and productivity is lower than other modern methods. Roof and side falls are the major hazards in underground bord and pillar mining method in India. The statistics show that about 61% of underground accidents are due to roof and side fall, accounting for 22% of total fatalities (DGMS Annual report, 2007). Bord and pillar method of working is carried in two stages, development and depillaring with total extraction of 50-60%. 20-30% coal is recovered during development of galleries.

Design of systematic support system is essential for providing safe working condition and to avoid roof and side fall accident.

Three diverse methodologies have been used for the design of systematic support:

- **Empirical approach**
- **Numerical modeling**
- **Observational approach**

The three methodologies were followed and compared to design systematic support for underground bord and pillar working of 1A-seam, RK-6 Inc, SCCL.

Empirical modeling is carried out by CMRI- RMR and NGI-Q Systems for formulating design of support in rock engineering (Bieniawski, 1976). CMRI-RMR system is used for design of support system in galleries during development stage and NGI-Q system is used for design of support during depillaring.

Numerical model of the seam condition was simulated to design systematic support for roof and side by rock bolt. Analysis of convergence of the galleries and stress re-distribution over pillar and stooks, for different stages of depillaring in underground bord and pillar working is also simulated using FLAC 2D software.

Field observations of the roof convergence of the galleries were regularly measured by trained SCCL personnel. The readings were recorded from the convergence station with the help of telescopic convergence rod. Convergence stations were installed at every 20 m interval in the galleries.

The analytical result was used in the field and also simulated in numerical modeling. The field data was analyzed and compared with numerical modeling results to validate the model for forecasting the behavior of strata in future workings.

1.1 Objectives of the Project

- To design systematic support for development and depillaring based on conventional empirical method.
- To evaluate deformation of the galleries and stress re-distribution over pillar and stooks at different stages of depillaring using the numerical modeling software FLAC 2D.
- To validate the model results with field observation data to provide suitable design guidelines for strata control in advance.

CHAPTER 2

LITERATURE REVIEW

LITERATURE REVIEW

Bord and pillar method is the most widely practiced underground coal mining method contributing over 90% of the underground coal working today. Bord and pillar method is suitable for flat deposits with thin seam, but its safety and productivity is lower than other modern methods. Roof and side falls are the major hazards in underground bord and pillar mining method in India. The complexity of geological deposit and variability of mining parameters leads to the occurrences of unwanted roof falls. Bord and pillar method of working is carried in two stages, development and depillaring with total extraction of 50-60%. The work carried out by many researchers has been reviewed and their inferences are shown below:

Singh et al. (2005)**: B**ord and pillar mining is very much in practice in Indian underground coal mines. Basically two empirical methods; CMRI-RMR for design of supports during development and Q-System for design of supports during depillaring are being used in India. The supports include full column grouted bolts, props and chocks.

Maiti et al. (2006)**:** Information on magnitude and direction of in-situ and induced stress is critical for safe design of underground workings. Numerical modeling is the preeminent advancement for solving and understanding strata control problems.

Cambulat (2008): Though roof bolting is prominent in use, the roof falls and strata control poses a major challenge. This is due to inherent uncertainties in rock mass and support elements which are not considered in design methodologies.

Palei and Das (2008): Calculation of support safety factor is important for support planning and design of underground coal mines for prediction of roof fall. The study infers that the gallery width is ranked as the first parameter to control the support factor of safety.

Maiti and Khanzode (2009): A relative risk model for roof and side fall accidents was developed by using log linear analysis of two way contingency table. The application reveals that effectiveness of safety measures across different locations in underground mines varies and focuses mainly in workplaces such as face.

Das et al. (2009): Their work predicts the severity of roof fall accidents. Their work inferred that unsupported or partially supported roofs are more prone to major as well as serious accidents and deep workings have higher risk of major accidents than the shallow workings.

Cambulat (2010): Advanced roof support design based on stochastic modeling technique ensures greater stability of roadways. The input parameters of stochastic modeling are taken as probability distribution rather than single values.

Kushwaha et al. (2010): A comprehensive guideline is developed for depillaring considering split and slice width, rock characterization, depth cover and in-situ stresses. Vibrating wire stress meters and strain gauged rock bolts were proven beneficial for their study in strata behavior and to make the decision of the amount of support to be provided.

Singh et al. (2011): Strong and massive roof strata provides stability during primary development but poses more problems during depillaring. The problem is more complex for deep workings.

Singh et al.(2011): The assessment of stability of the three basic mining structures, i.e. pillar, roof strata, applied supports at different stages of an underground coal mining is important for optimization of safety and recovery is inferred from the study.

Singh et al. (2011): The in-situ and mining induced stresses has a greater impact on performance of bord and pillar mining. The in-situ stresses are generally static in nature where as mining induced stress vary over pillar and are highly influenced by strata dynamics during different stage of extraction.

Jayanthu et al. (2012)**:** Reexamination and modification of the norm for design of SSR in development is needed with consideration of life of the roadway. Understanding the strata behavior at critical stages of roof fall is required besides approaches for design of strata control techniques. Instrumentation is required for continuous monitoring of strata behavior in provisions of convergence of openings and stress over pillars and stooks in advance of the extraction line. Formulation of Strata Control Cell for designing Systematic Support Rule (SSR) and monitoring strata control measures in a scientific way is necessary to ensure efficacy.

2.1 Systematic Support Rules (SSR): 108 – CMR, 1957

The provisions of this regulation with respect to systematic support is applicable to –

a) Every district in a mine in which extraction or reduction of pillars is going on.
b) Every “longwall” working - every development working within 10 meters of face and every junction of roadways immediately out-bye of a development face.
c) Every working in a disturbed or crushed ground and

d) Any mine or part of a mine where, in the opinion of the regional inspector, the roof or side is of such a nature as to required artificial support.

The manager of every mine having workings below ground shall, before commencing any operation specified above and also when required by the Regional Inspector, frame, with due regard to the physio-mechanical properties of strata, local geological conditions, system of work and mechanization, and past experience, and enforce Systematic Support Rules specifying in relation to each working place the type and specifications of supports and the intervals between.

(i) Supports on roadways where machinery is used for cutting, conveying or loading.
(ii) Each row of props, roof bolts or other supports.
(iii) Adjacent rows of props, roof bolts or other supports.
(iv) Last row of supports and the face.
(v) Hydraulic chocks and powered supports and
(vi) The pack and the face.

2.1.1 DGMS - Guidelines for Support System

The guidelines for support system circulated by DGMS states:

a. In the beginning of the shift, the support man accompanied by the Mining sirdar shall check and test the conventional supports in the development and depillaring area and shall assess the requirement of additional supports to be provided during the shift.
b. After every round of blasting, support of roof and sides shall be tested at all places within the zone of influence of blasting as decided by the manager.
c. Bolting of roof shall be done as soon as possible after exposure of the roof.
d. 9% of the bolts shall be subjected to anchorage testing for assessment of prescribed anchorage strength and 1 percent shall be subjected to destructive testing to assess the efficacy of support requirement and a record of such tests shall be maintained in a bound paged register kept for the purpose in the format as prescribed by D.G’s Technical Circular No.3 of 1996.
e. Separate crew shall be provided for the haulage and the traveling roadways and old workings.
f. Where the coal has a tendency to spell, the sides shall be kept supported systematically in addition to roof supports.

2.2 CMRI- Rock Mass Classification (RMR) - ISM

This rock mass classification system is being used by industry, academicians and research institutes. The five parameters used in the classification system and their relative ratings are summarized in Table 2.1.

Table 2.1: CMRI-RMR Determination Parameters

Sl. No	Parameter	Max. rating
1	Layer thickness	30
2	Structural features	25
3	Rock weatherability	20
4	Strength of roof rock	15
5	Ground water seepage	10

The Rock Mass Rating system is presented in Table 2.1, giving the ratings for each of the five parameters listed above. These ratings are summed to give a value of *RMR*. Determination of the parameters is done individually for different layers of the rock types in the roof up to a height of at least 2 m.

Rock Mass Rating (RMR) is the sum of five parameter ratings. For more than one rock range in the roof, RMR is calculated separately for each rock type and the collective RMR is obtained as:

$$\textbf{Combined RMR} = \frac{\Sigma\ \textbf{(RMR of each bed x bed thickness)}}{\Sigma\ \textbf{(Thickness of each bed)}}$$

The RMR obtained may be adjusted if necessary to take account for some special situations in the mine like depth, stress, method of work.

In applying this classification system, the rock mass is divided into a number of structural regions and each region is classified separately. The boundaries of the structural regions usually coincide with a major structural feature such as a fault or with a change in rock type. In some cases, significant changes in discontinuity spacing or characteristics, within the same rock type, may necessitate the division of the rock mass into a number of small structural regions.

Table 2.2: RMR Classifications

Sl No	Rock mass rating	Rock quality
1	0 – 20	Very poor
2	20 – 40	Poor
3	40 – 60	Fair
4	60 – 80	Good
5	80 – 100	Very good

2.3 Q-Classification of Rock Mass

Q-Classification of Rock Mass is required to design support in a depillaring panel with widely varying geo mining conditions corresponding to different support density. Support design for a conventional depillaring area is determined by Barton's Rock mass classification index- Q. The parameters *Jn*, *Jr* and *Ja* appear to play a more important role than orientation, because the number of joint sets determines the degree of freedom for block movement (if any), and the frictional and dilation characteristics can vary more than the down-dip gravitational component of un-favorably oriented joints. If joint orientations had been included the classification would have been less general, and its essential simplicity lost. The rock quality Q-system can be considered to be a function of three parameters which are crude measures of:

1. Block size (*RQD/Jn*)
2. Inter-block shear strength (*Jr/ Ja*)
3. Active stress (*Jw/SRF*)

The **block size** (RQD/Jn), representing the structure of the rock mass, is a simple gauge of the block or element size, with the two extreme values (100/0.5 and 10/20) differing by a factor of 400.

The **inter-block shear strength** (*Jr/Ja*) represents the roughness and frictional characteristics of the joint walls and filling materials. This proportion is weighted in favor of rough, unaltered joints in through contact.

The **active stress** (*Jw/SRF*) consists of two stress parameters. *SRF* is a gauge of loosening load of an excavation through shear zones and clay bearing rock. SRF measures rock stress in competent rock and plastic incompetent rocks. It can be used as a total stress parameter.

Q = (RQD/Jn)*(Jr/Ja)*(Jw/SRF)

RQD=Rock Quality Designation = f (layer thickness) = 0 to 100

Jn=	Joints Set Number	= 0.5 to 20
Ja=	Joint Alteration Number	= 0.75 to 24
Jw=	Joint Water Reduction	= 0.005 to 1
Jr=	Joint Roughness Number	= 1 to 4

SRF= Stress Reduction Factor, It's value varies for a range of geometries during excavation are as follows:

	SRF
For galleries and junctions:	1-2
For slices:	2-5
For goaf edges:	10

Roof pressure is estimated by the relations based on the Q value accustomed to the geometrical conditions:

For joint set number (Jn)> 9, $\mathbf{P_{roof}}$(roof pressure) $= 2/Jr * (5Q)^{1/3}$

For Jn < 9, $\mathbf{P_{roof}}$ $= 2Jn^{1/2} /3\ Jr * (5Q)^{1/3}$

2.4 Numerical Modeling

Numerical modeling in geo-mechanics is used to understand the governing mechanisms affecting the behavior of the system. Once the behavior is understood, it is then appropriate to develop calculation for a design process. The finite difference method is perhaps the oldest numerical technique used for the solution of sets of differential equations. In the finite difference method, each derivative in the set of principal equations is replaced directly by an algebraic expression written in terms of the field variables at discrete points in space; these variables are undefined within elements. The formulation involves the adjustment of these parameters to minimize error terms. In Lagrangian formulation, incremental displacements are added to the coordinates so that the grid moves and deforms with the material it represents. FLAC (Fast Lagrangian Analysis of Continua) uses an explicit, time marching method to solve the algebraic equations. FLAC 5.0 has been used to simulate and analyze the field condition.

2.4.1 FLAC 5.0

FLAC (Fast Lagrangian Analysis of Continua) is an explicit finite difference program in two-dimension developed for engineering mechanics computation. The program is used to simulate the performance of structures that undergoes plastic flow when their yield limits are reached. Materials are represented by elements which form a grid that is adjusted by the user to fit the shape of the object to be modeled. Every grid behaves according to a arranged linear or nonlinear stress and strain law in reaction to the applied forces. The material can yield and flow and the grid can deform and move with the material that is represented. The explicit, Lagrangian calculation scheme and the mixed-discretization zoning technique used in FLAC ensure that plastic collapse and flow are modeled very accurately. Because no matrices are formed, large two-dimensional calculations can be made without excessive memory requirements. The drawbacks of the explicit formulation are overcome to some extent by automatic inertia scaling and automatic damping that do not influence the mode of failure. FLAC has various built-in material models:

- The “null” model, which represents excavations in the grid;
- The isotropic elastic model;
- The transversely isotropic elastic model; and
- Eleven plasticity models (Drucker-Prager, Mohr-Coulomb, ubiquitous-joint, strain-hardening/softening, bilinear strain hardening/ softening ubiquitous-joint, double-yield, Hoek-Brown, modified Hoek-Brown, modified Cam-clay, cap-yield soil model and simplified cap-yield soil model).

FLAC can also be used to create constitutive models by using the FISH programming language. Each zone in a FLAC grid may have a different material model or property, and a continuous gradient or statistical distribution of any property may be specified. FLAC contains many special features, including

- Interface elements to simulate distinct planes along which slip or separation can occur;
- Plane-strain, plane-stress and axis symmetric geometry modes;
- Groundwater and consolidation models with automatic phreatic surface calculation;

- Structural element models to simulate structural support;
- Automatic re-meshing logic to generate a regular mesh, and prevent a badly distorted grid, during the solution process in large strain simulations;
- “Virtual-grid” generation tools available through a graphical-user interface to facilitate model construction;

2.4.2 Procedure Recommended for numerical modeling

The procedure recommended for solving a real life situation can be enlisted as below:

Task 1 Define objectives for the model analysis

Task 2 Model a conceptual picture of the geological system

Task 3 Simulate simple idealized models

Task 4 Assemble problem-specific data

Task 5 Simulate series of detailed model runs

Task6 Calculate model performances

Task 7 Interpret the results

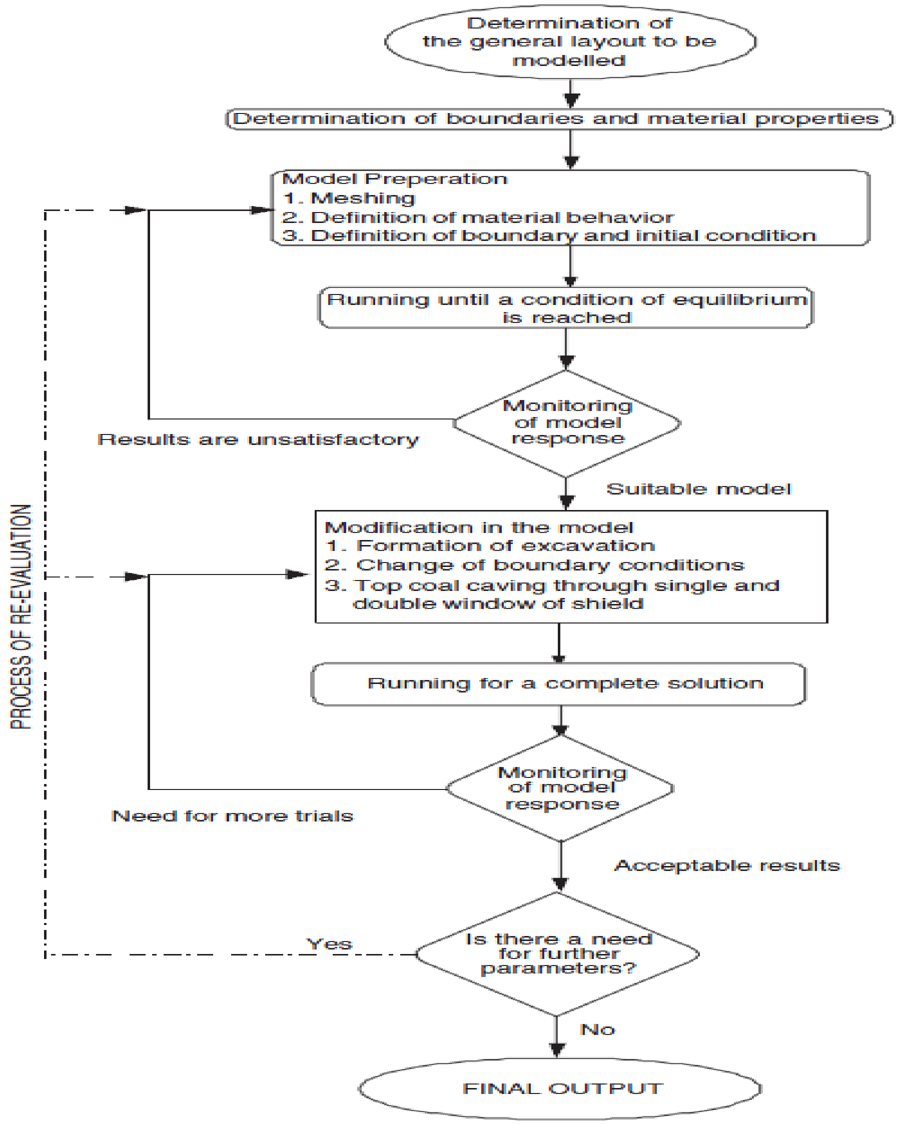


Figure 2.1: A General Flow Chart for Numerical Modeling

2.5 Instrumentation and Monitoring

Instrumentation is carried out in the galleries for monitoring various parameters of strata behavior. Locations of strata behavior monitoring stations commissioned in the panel are chosen scientifically. The following instruments are used to monitor the strata behavior.

a) Convergence stations	- Telescopic rod type
b) Load cells	- Electronic type
	- Mechanical type
c) Stress meters	- Vibrating wire type
d) Extensometers	- Tell-tale (four-point) type

The **telescopic rod convergence indicator** is a simple instrument consisting of a graduated rod fitted in a pipe. It has a least count of 0.5 to 1 mm, and the telescopic movement is for a length of 2 to 4 m. The measuring points are metal rods grouted in the roof and floor.

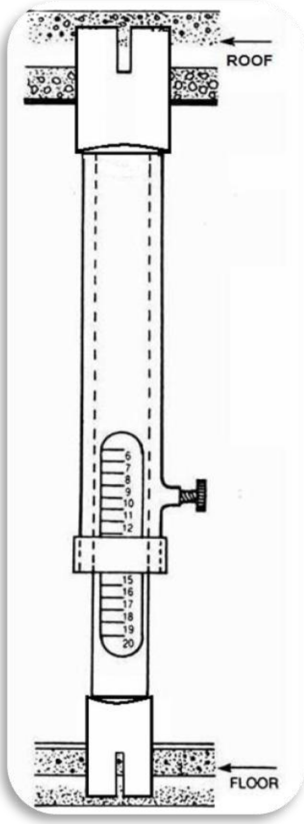


Figure 2.2: Telescopic Convergence Rod

Measurements are taken by simply stretching the telescopic rod between the reference points, and reading the graduations on the rod. Convergence stations were installed at every 20 m interval in the levels of the panel. These indicators are useful for understanding the roof to floor closure in the advance galleries at various stages of extraction. Rate of the closure may give some indication of the impending roof falls.

The **electronic load cells** work on the principle of vibrating wire gauge. The vibrating wire gauge consists of a stretched wire, which is plucked by a pulse of high energy. Changes in the load exerted on the cell cause changes in the length of this wire, resulting in variations of frequency of vibration. This frequency is measured by a digital read-out unit, and is converted into load using calibration charts. The load cells were installed under the hydraulic props using specially prepared steel seating arrangement.

Figure 2.3: Electronic Load Cell

In **mechanical load cells**, a dial gauge is used for measurement of the compression of a spring. The amount of compression is converted into load using a calibration chart for the respective load cells. Efficacy and adequacy of the present support system can be inferred on the basis of these load cells.

Figure 2.4: Mechanical Load Cell

The typical four-point wire type bore hole extensometer, known as "**Tell-Tale instrument**" consists of four spring anchors, steel wires, four position indicators and a reference tube. The anchors are fixed inside the bore hole of 40 mm diameter at different horizons, namely, 1 m, 2 m, 4 m and 6 m in the hole. After inserting the anchors, the free end of the wire is passed through the reference tube and crimped to the indicators. Reading methods are based on colour and also the scale marked for each anchor. Close observation of the scale on the indicator tube would give the amount of de-lamination with 1 mm accuracy. Movement of the indicator with green colour relative to its reference is equal to the strata separation in the roof, that is, up to the first anchor position. Bed separation between the four anchor positions can be accessed from the reading on the other three indicators. However, the bed separation above the top anchor is not measurable with this instrument. These instruments may also be useful for inferring the effectiveness of the support system in eliminating and minimizing the tendency of bed separation in the roof.

Figure 2.5: Tell-Tale Extensometer

The vibrating-wire **stress meter** is used for measuring unidirectional stress change in coal/rock. It consists essentially of a wire tensioned across a steel cylinder. As the rock/coal stress changes, the cylinder deform, causing the tension in the wire to change. A bore hole of 42 mm diameter is required for installing stress meters, preferably at mid height of the pillar either horizontally or slightly rising or dipping according to dip of the seam. Stress meter along with wedge and platen assembly is set in the borehole, at a depth of about 3.5 m. The trend of variation of stress over pillar or stooks may indicate the extent of abutment loading in advance of the line of extraction.

Figure 2.6: Vibrating-Wire Stress Meter

Instruments used for monitoring of strata behavior in the field are compared with their application, cost, advantage and disadvantage.

Table 2.3: Comparison of Different Instruments

Instruments	Application	Cost	Advantage	Disadvantage
Telescopic Convergence Rod	To measure convergence of galleries	Rs. 3,900	Easy to use	Not Accurate
Electronic Load Cell	To measure load on supports	Rs. 12,000	Continuous monitoring	High cost
Mechanical Load Cell	To measure load on supports	Rs. 5,500	Direct display	Manual reading
Tell-Tale Extensometer	To measure bed separation	Rs. 12,000	Multilayer monitoring	Manual reading
Vibrating Wire Stress Meter	To measure stress change	Rs. 80,000	Auto data logging	High cost

CHAPTER 3

METHODOLOGY

METHODOLOGY

3.1 Design of Support System

3.1.1 Estimation of Rock Load and Design of Support System for Galleries and Splits

Rock load in the galleries and splits in depillaring areas was determined using the empirical relationship of CMRI-RMR System.

Table 3.1: RMR Calculation

Sl. No	Parameter	Description	Rating
1	Layer thickness	15cm	17
2	Structural features	Joint Slip	13
3	Rock weatherability	91%	10
4	Strength of roof rock	195 Kg/cm^2	5
5	Ground water seepage	Moist	7
	Total RMR		52(Fair Rock)

***Rock load = B × D* (1.7-0.037 × *RMR* + 0.0002 × *RMR*^{2})**

RMR = 52

Gallery span (B, width) = 4.2 m

Height = 3m

Density (D) = 2.1t/m^2

Rock load = 2.75 t/m^2

Hence, rock load in galleries and splits = 2.75 t/m^2

The rock load is to be supported with higher load to protect the area from the roof fall. Since the safety factor is used in equation, therefore the roof support is designed for rock load of2.75 t/m^2.Varioustypes and capacity of supports are available for design of Support system in bord and pillar workings. Full column grouted bolt is used as the support system.

Roof bolt support = 10 t

Bolt spacing = 1.5 m

Distance between two rows of bolts = 1.4 m

No of bolts in a row= 3

Support resistance = 30 t/5.4m^2 **= 5.1 t/m^2**

Factor of safety = 5.56 / 2.75 = **2.02** for gallery.

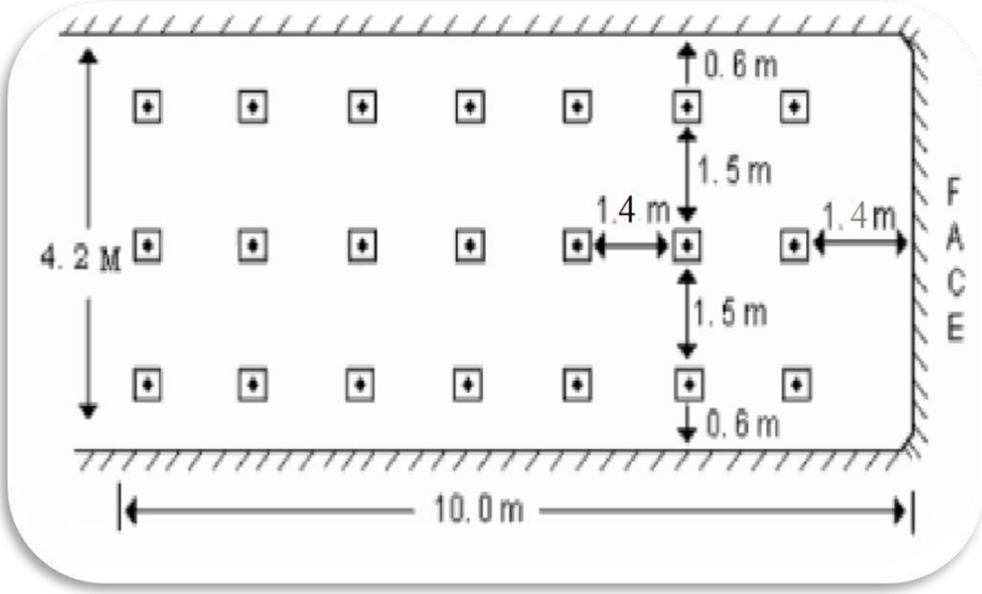


Figure 3.1: Design of Support System for Galleries and Splits

3.1.2 Design of Support at Junctions

The junctions are unstable and experience more load when compared with galleries due to large surface area. The total area of the junction is 4.5mx4.5m. Therefore the junctions are supported by are Supported by 33% extra bolts.

Rock Load= 4.2*4.2*2.75 = 60.75 t

Support Resistance = 13*10 = 130 t

Factor of safety = 130/60.75 = 2.13

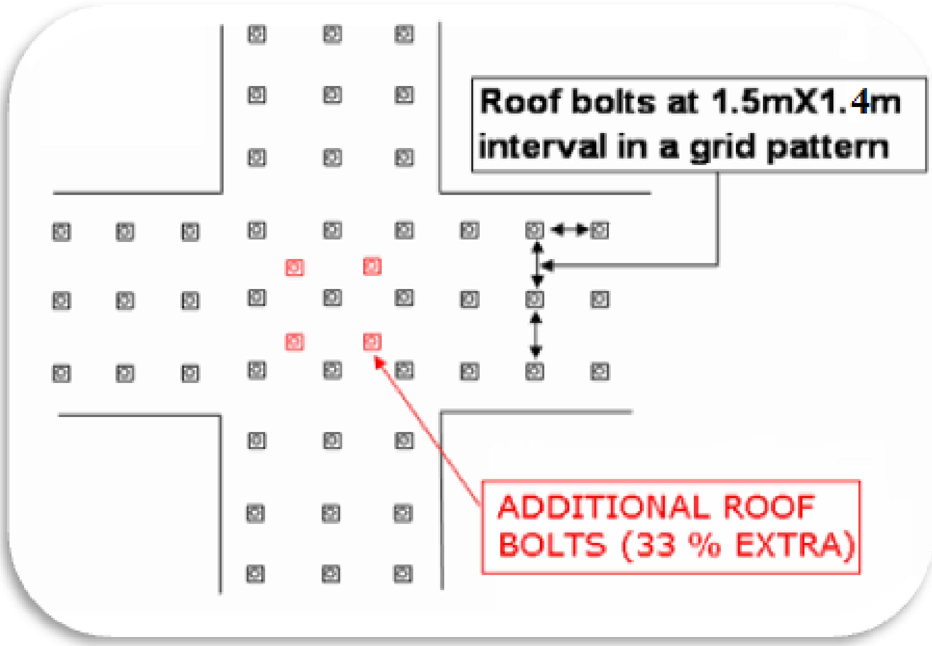


Figure 3.2: Design of Support System at Junctions

3.1.3 Estimation of Rock Load and Design of Support System in Depillaring Working

Rock load (P_{roof}) in slice and goaf edge was estimated using NGI-Q system from the following empirical relation:

$$P_{roof} = 2/3\ (Jn1/2/\ Jr)\ x\ (5Q)^{-1/3}$$

Where, Jn = 9, Jr = 1.5, Q = 2 for slice and Q = 1 for goaf edge.

Table 3.2: Q-System Calculation

Parameters	RQD	Jn	Ja	Jr	Jw	SRF		Q	
						Slices	Goaf Edges	Slices	Goaf Edges
Values	60	9	1	1.5	1	5	10	2	1

The rock load in the slice is calculated to be (P_{roof}) = 6.19 t/m^2 and rock load at goaf edge is calculated to be P_{roof}= 7.79 t/m^2. The slice and goaf edges are supported by steel props and chocks.

Slice width = 4m

Rock load in slice, P_{roof} is 6.19 t/m^2

Breaker line bolt support = 8 t

Chock with corner prop support = 30 t

The support system will be three chocks with corner prop and five breaker line bolt as shown in Figure3.3. The above configuration leads to:

Support resistance =130 t/12sq.m **= 10.5 t/ m^2**

Factor of safety = 10.8/6.19 = **1.75** for slices

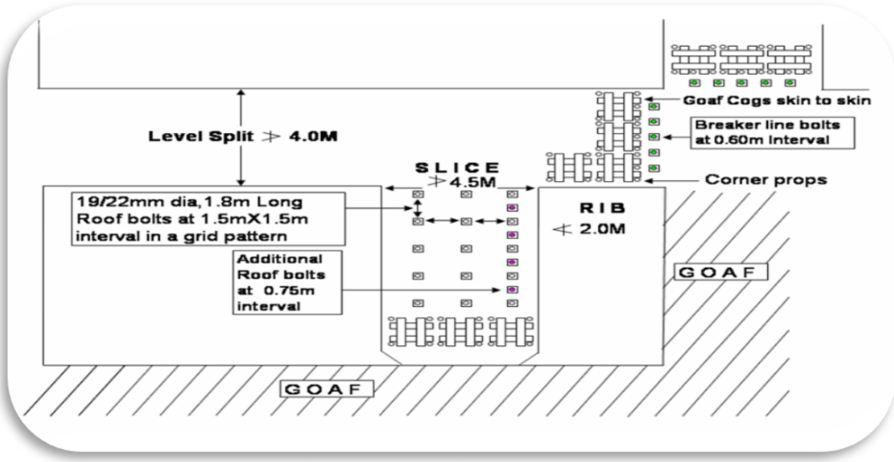


Figure 3.3: Design of Support System for Slices

Goaf edge side will also have 3 chocks with corner prop and5 breaker line bolts with spacing of 0.8m. The other calculations are as follows:

The support resistance = 130 / 12= **10.8 t/m^2**

Factor of safety = Roof support / Rock load for goaf edge = 10.8 /7.79 = **1.39**

It is good because goaf edge is supported for temporary period.

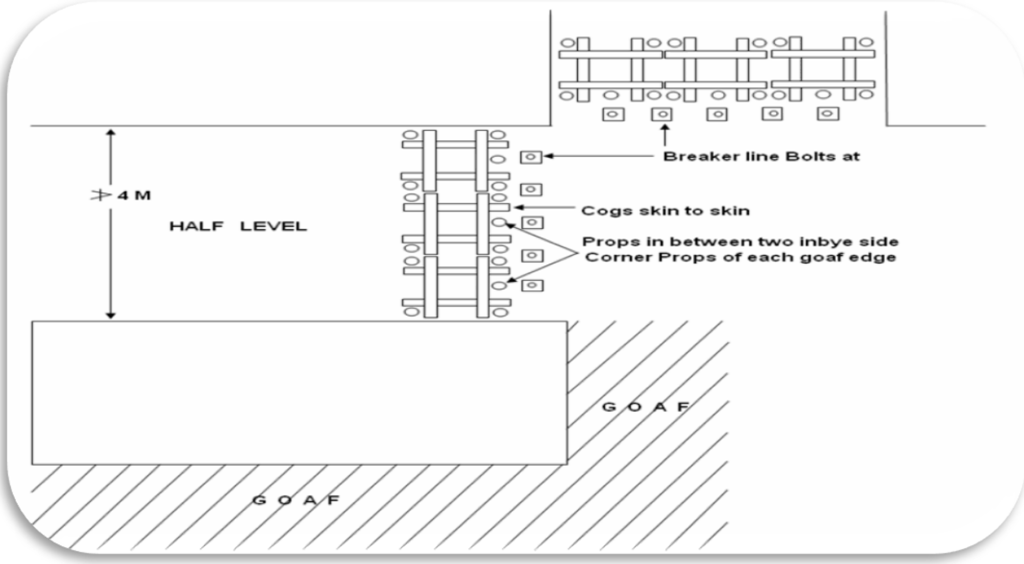


Figure 3.4: Design of Support System for Goaf Edges

3.1.4 Design of Support System for Sides of Galleries

Presence of 0.3m thick clay band, 1m above the floor of the seam reduces the stability of the sides. Therefore the sides of the galleries are supported by stitching the wire ropes with the rock bolt. Three rock bolts are installed in a row with 1m spacing and distance between the rows is 1.5m. Wooden lagging are provided at 1.5m interval to increase the contact area of the stitching rope and to provide better support.

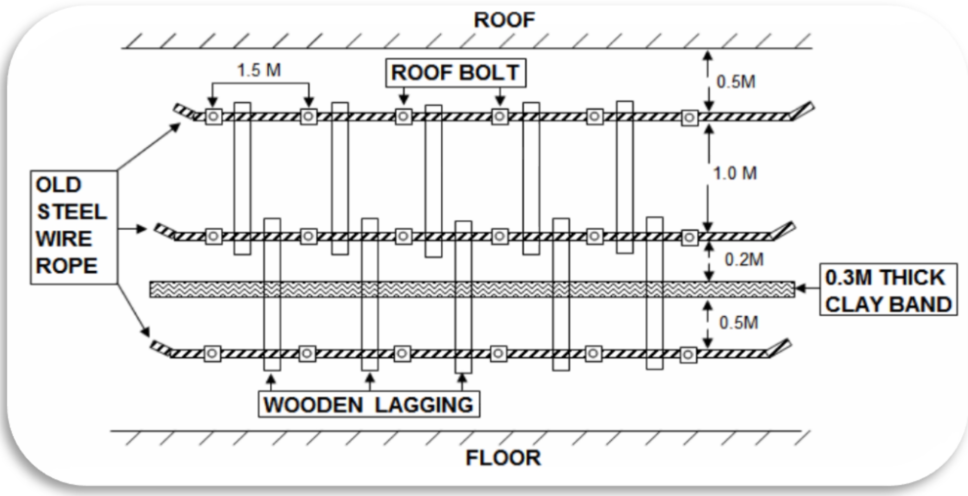


Figure 3.5: Design of Support System for Sides

3.2 Numerical Simulation

The numerical simulation of the geo-mining condition of 1A seam of RK-6 Inc is done by generating models. The geo-mining details of the RK-6 Inc are shown in Table 3.3.

Table 3.3: Geo-Mining Details of RK-6 Inc Mine

Total Thickness	5.50m
Gallery size	4.5 m * 3m
Pillar size	30m * 30m (centre to centre)
Depth of Working (average)	210 m
Nature of roof and floor	Grey sand stone
Compressive strength of coal	360 kg/cm^2
RMR	52.2 (fair roof)
Water seepage	100 - 200 ml./ min
Full dip gradient	1 in 4
Apparent dip gradient	1 in 7
Grade of the coal	F

The pillar size and the dimension of the gallery considered in the model are 30 m and 4.2mx3m respectively. After the generation of development model in first stage the pillars were given splits of 4 m and effects were studied. In later stages the seams was extracted in stages. Ribs were left in the goafs. Numerical modeling was then used to study the convergence and stress conditions in the pillars in development stage, in stooks and ribs.

3.2.1 Design Parameters of the Model

The elements in the panel considered are small. The elements are of size 1 m vertically and 1 m horizontally in the pillars. The dimensions of mesh elements increase geometrically from the inner model to the outer boundary. This is done because accurate reading over the seam is only required. Varying mesh size also reduces the simulation and computation time of model as the elements at the boundary were of greater dimension. The development model is then modified into an excavation model. Mohr Coulomb criteria and plain strain condition are used for simulation of the model. The sandstone element was used as the depth covers and the floor material.

Table 3.4: Parameters used in the Numerical Modeling

Property	**Coal**	**Sandstone**	**Clay Band**
Bulk Modulus	3.67 GPa	6.67 GPa	2 GPa
Shear Modulus	2.2 GPa	4.0 GPa	1.4 GPa
Density	1480 kg/m^3	2100 kg/m^3	1650kg/m^3
Tensile Strength	1.86 MPa	9.0 MPa	6000 Pa
Cohesion	1.85 MPa	6.75 MPa	5000 Pa
Friction Angle	30^0	45^0	17^0

The top edge of the model is unconstrained and allowed to move in any direction. The side edges of the model are constrained to move in x direction and left free to move in y direction. The bottom edge of the model is constrained in moving in y direction that is vertically. The in-situ vertical and horizontal stresses were calculated:

Vertical stress = ρ x H

Horizontal stress = 3.75 + 0.015 H (Kushwaha et al., 2010)

Where, ρ = specific gravity of the rock mass cover and H = depth of cover.

The model is simulated to generate the in-situ stresses, before adding the mine openings or galleries to the model. Then the mine opening or galleries required are added to the model. After this the simulation is re-simulated to give the final displacement and stress distribution.

3.2.2 Numerical Modeling Design

The 3 pillars have been modeled using FLAC5.0 with 4 galleries. At galleries three roof bolts are provided for supporting the roof strata and three bolts for supporting the sides. The supported galleries are simulated to plot their vertical displacement and vertical stress contours over pillars. It shows different stages of a depillaring process. The different stages include division of pillars, splitting, and extraction of the stooks so formed leaving just ribs in the goafs. Program code for numerical model is given in Annexure 2. The sequence of Numerical modeling includes the following stages:

Stage 1. Galleries and pillars are developed in the seams.

Stage 2. Splits and stooks are developed in three pillars.

Stage 3. One Stook was extracted and convergence readings were noted for each gallery.

Stage 4. One Pillar was extracted and convergence readings were noted for each gallery.

Stage 5. Three stooks were extracted and convergence readings were noted for each gallery.

Stage 6.Two pillars were extracted and convergence readings were noted for each gallery.

Stage 7. Five stooks were extracted and convergence readings were noted for each gallery.

Grid generated to simulate the model was presented below for different stages of extraction.

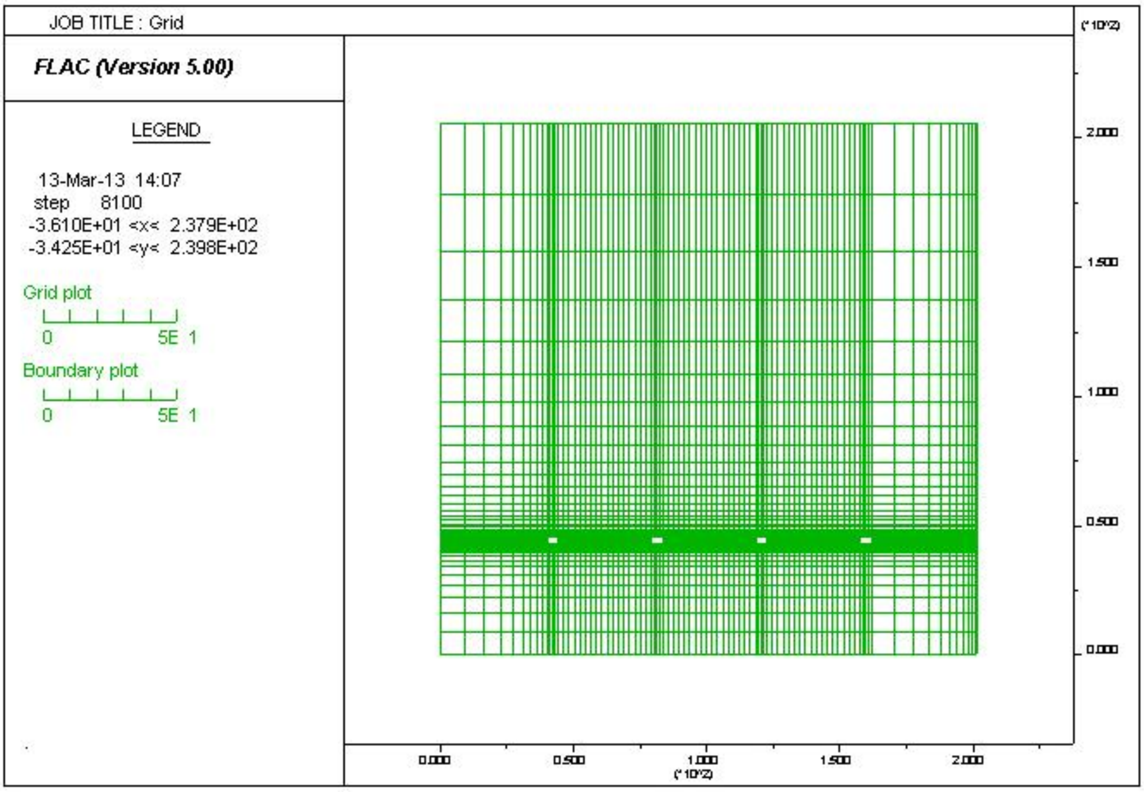


Figure 3.6: Grid Generation for Development of Galleries and Pillars in the Seams

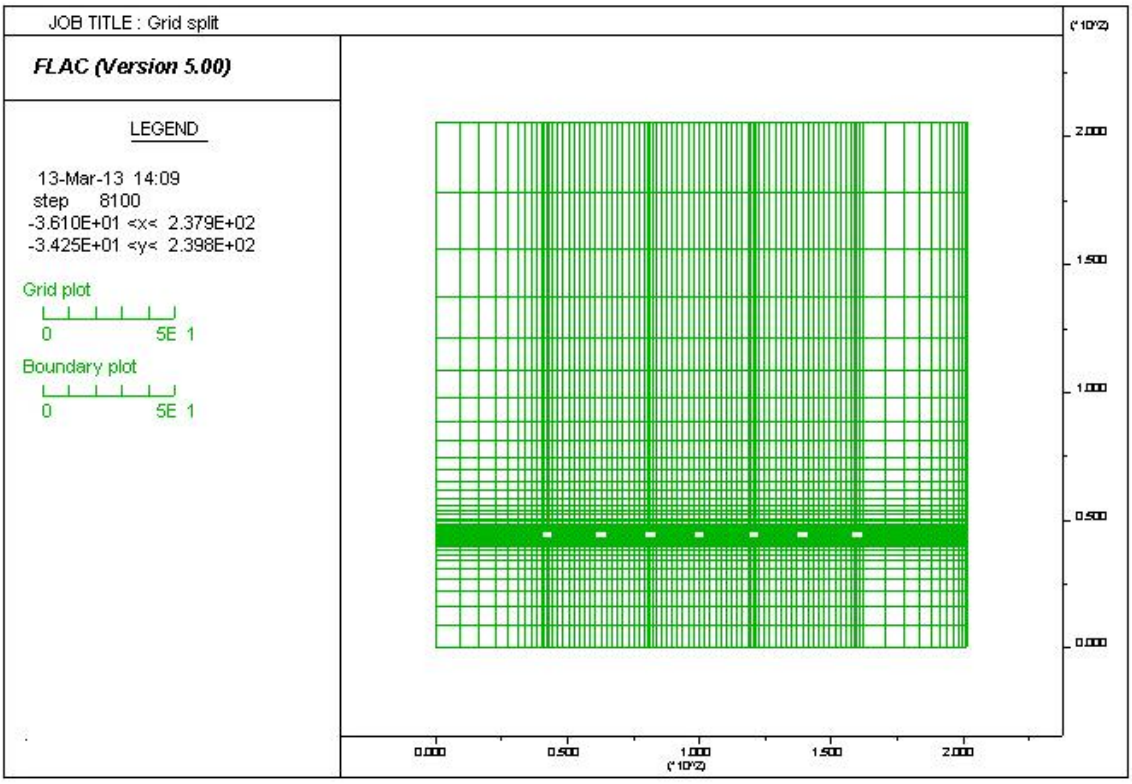


Figure 3.7: Grid Generation for Development of Splits and Stooks in three Pillars

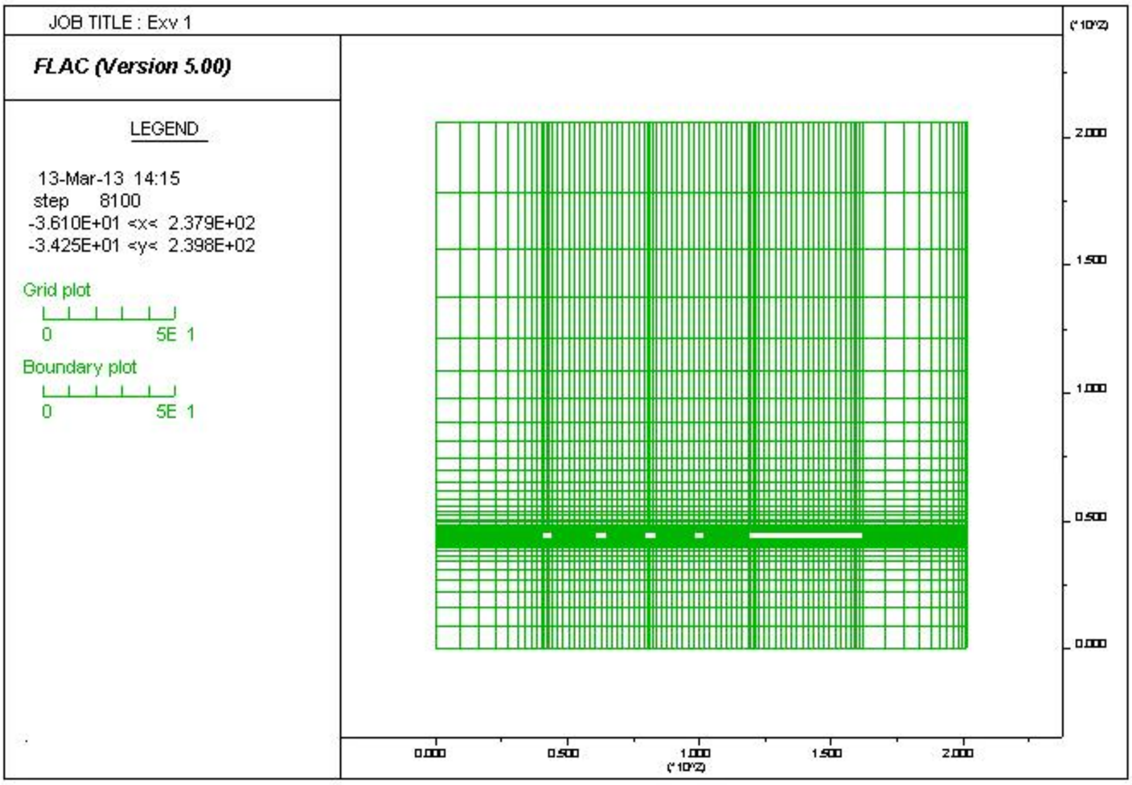


Figure 3.8: Grid Generation for Extraction of one Pillar

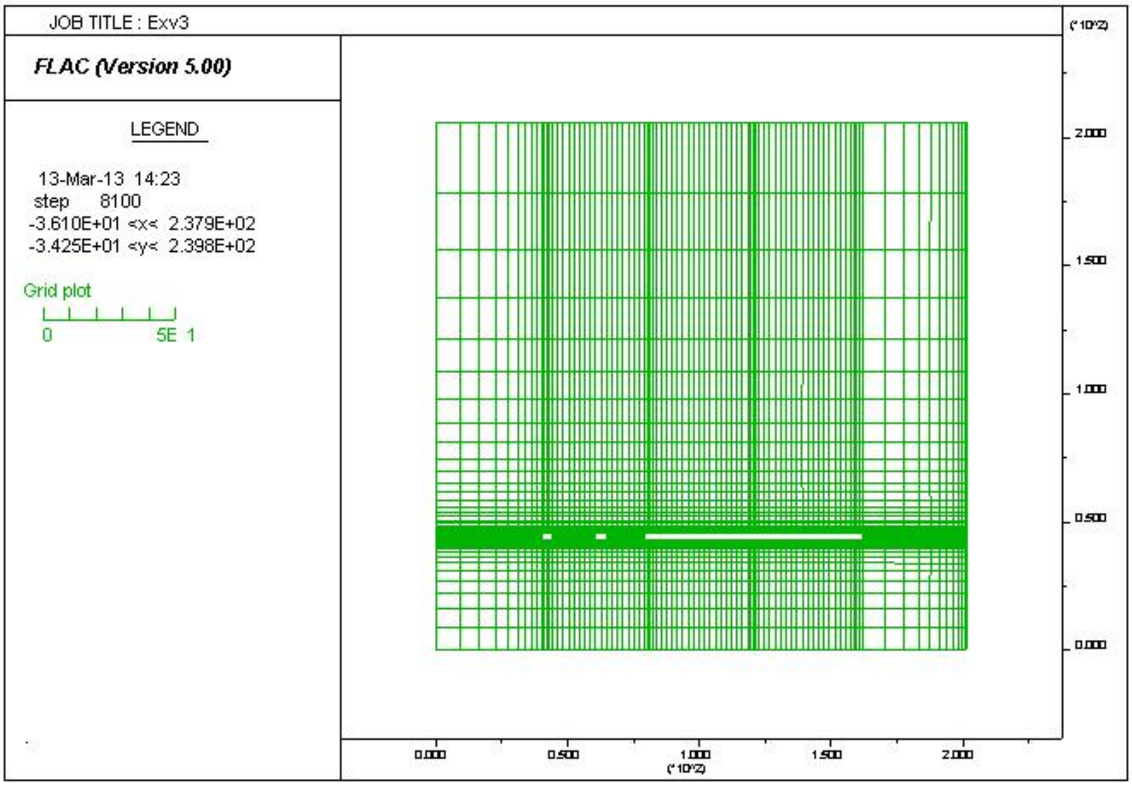


Figure 3.9: Grid Generation for Extraction of two pillars

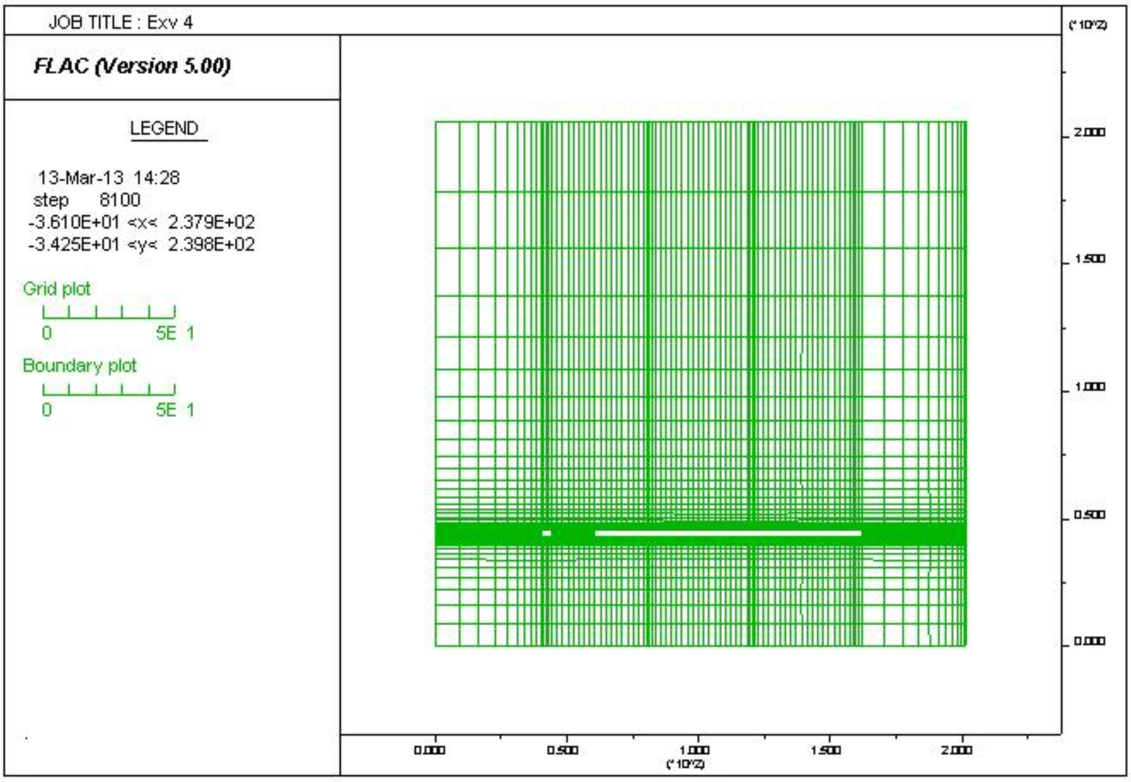


Figure 3.10: Grid Generation for Extraction of five Stooks

3.3 Field Monitoring at RK-6 Inc

To understand the behavior of the strata, it was considered necessary to conduct strata control investigations. It is well established that the behavior of the strata in depillaring can only be properly understood by measuring the diagnostic parameters such as convergence in galleries. As a part of this thesis, these quantities were measured regularly for a period of eight months or till the completion of depillaring of the panel using modern, proven instruments and techniques such as telescopic convergence rods. In addition, observations of the physical condition of the face were made regularly.

3.3.1 Instrumentation at 1AS2 Panel

Convergence recording stations were installed at all junctions situated within two pillar distance from pillar under extraction in the proposed panel. More than 35 Convergence stations were installed throughout the panel. Telescopic convergence rod was used to measure the daily convergence at convergence stations.

Instrumentation layout for strata monitoring in the panel No.1A S2was made in consultation with the mine authorities.

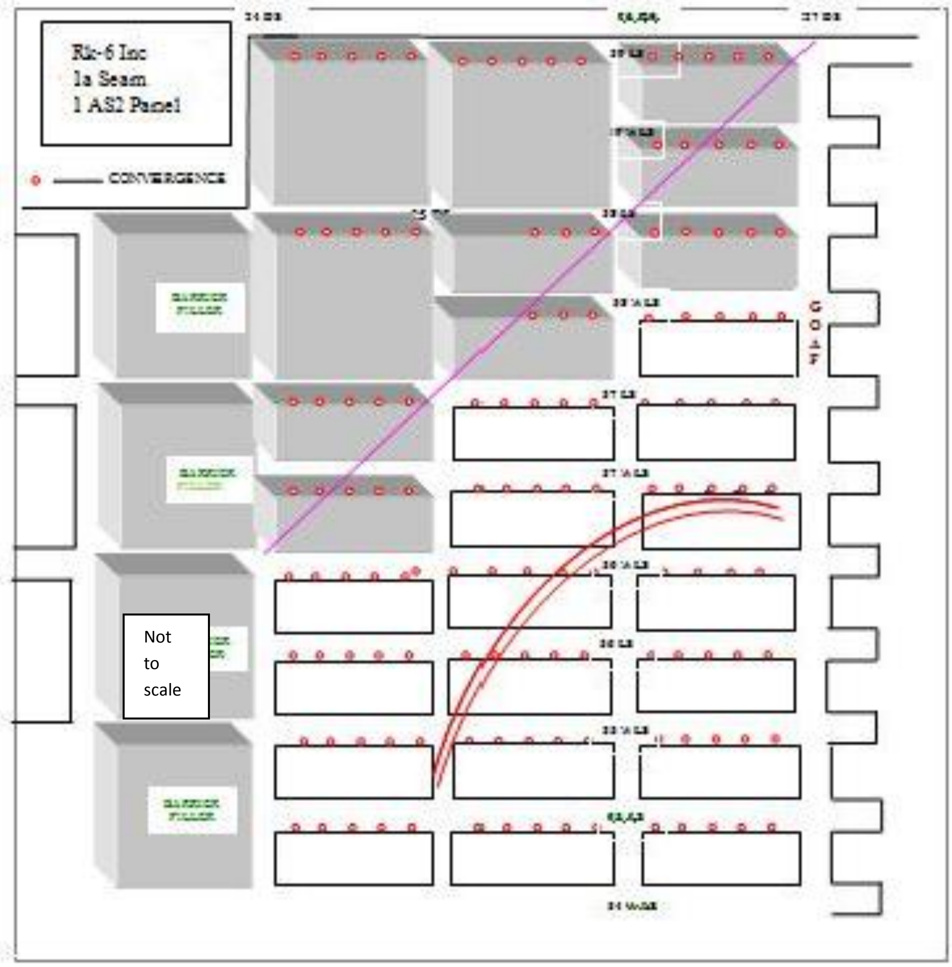


Figure 3.11: Instrumentation Layout of 1A S2 Panel of RK6 Incline

Installation of convergence stations was done at 10 m interval in the galleries along the levels and sublevels; 58½LS, 58LS, 57½LS, 57LS, 56½LS, 56LS, 55½LS, 55LS, and 54½LSin the 1AS2 panel of RK-6 Inc. The convergence stations were moved continuously with the approaching goaf and were installed within two pillars from the line of extraction.

3.3.2 Convergence Monitoring

Monitoring of readings at convergence recording stations were done in every shift by a competent person duly authorized by the manager with a telescopic convergence rod and the measurements were recorded in a bound paged book and the same were counter signed daily by Under Manager of the shift and Asst. Manager in charge. The data measured regularly in the 1AS2 panel is shown as comprehensive graphs below.

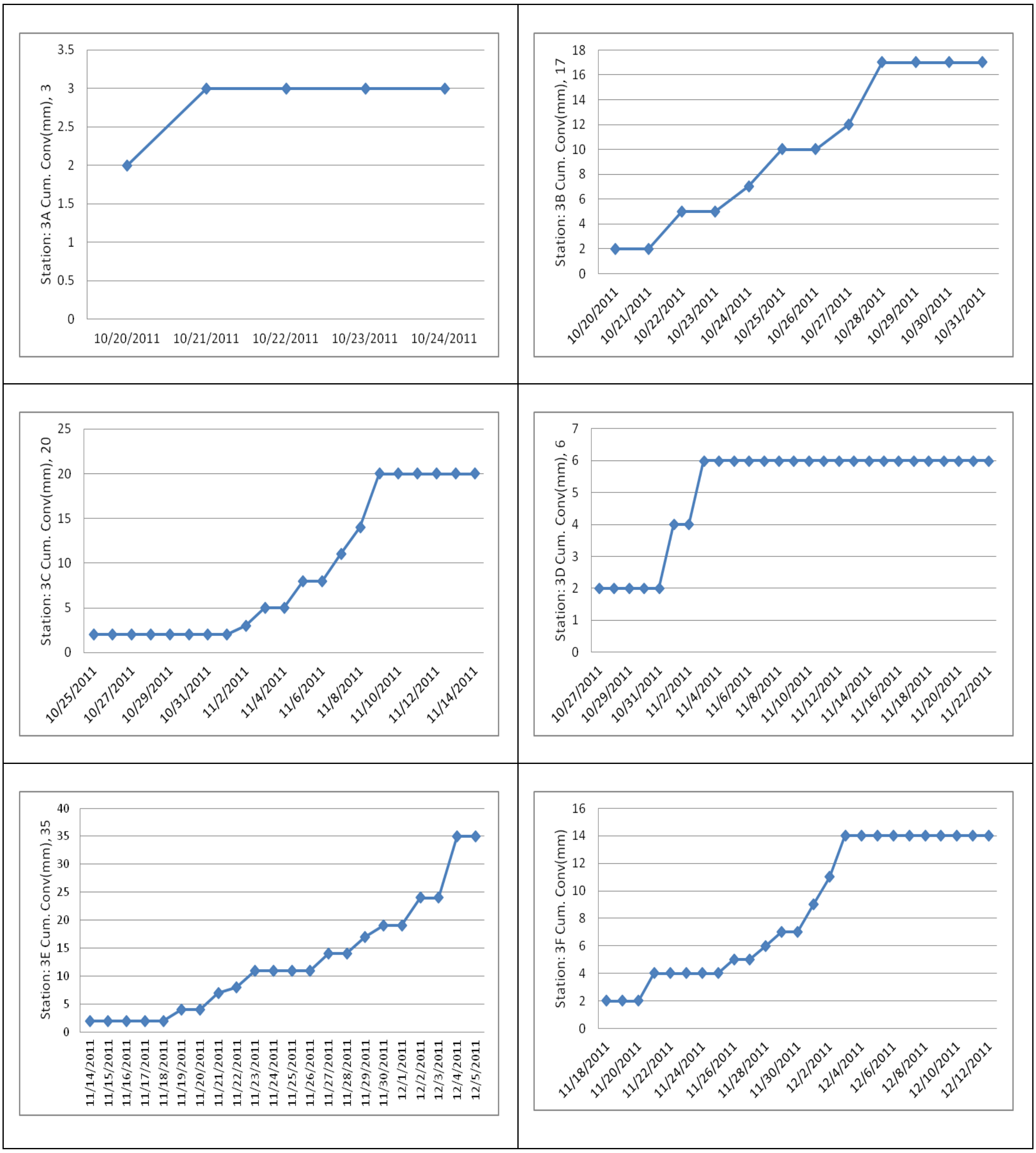


Figure 3.12: Convergence Observations in 58 ½ LS

Maximum convergence observed at level 58½LS was at station 3E installed on 14th October'2011 at a distance of about three stook from the goaf edge. The maximum daily convergence recorded was 7 mm when the goaf edge was 4 m from the station. Total cumulative convergence recorded at this station was 36 mm. Maximum convergence was observed when station was next to the goaf edge.

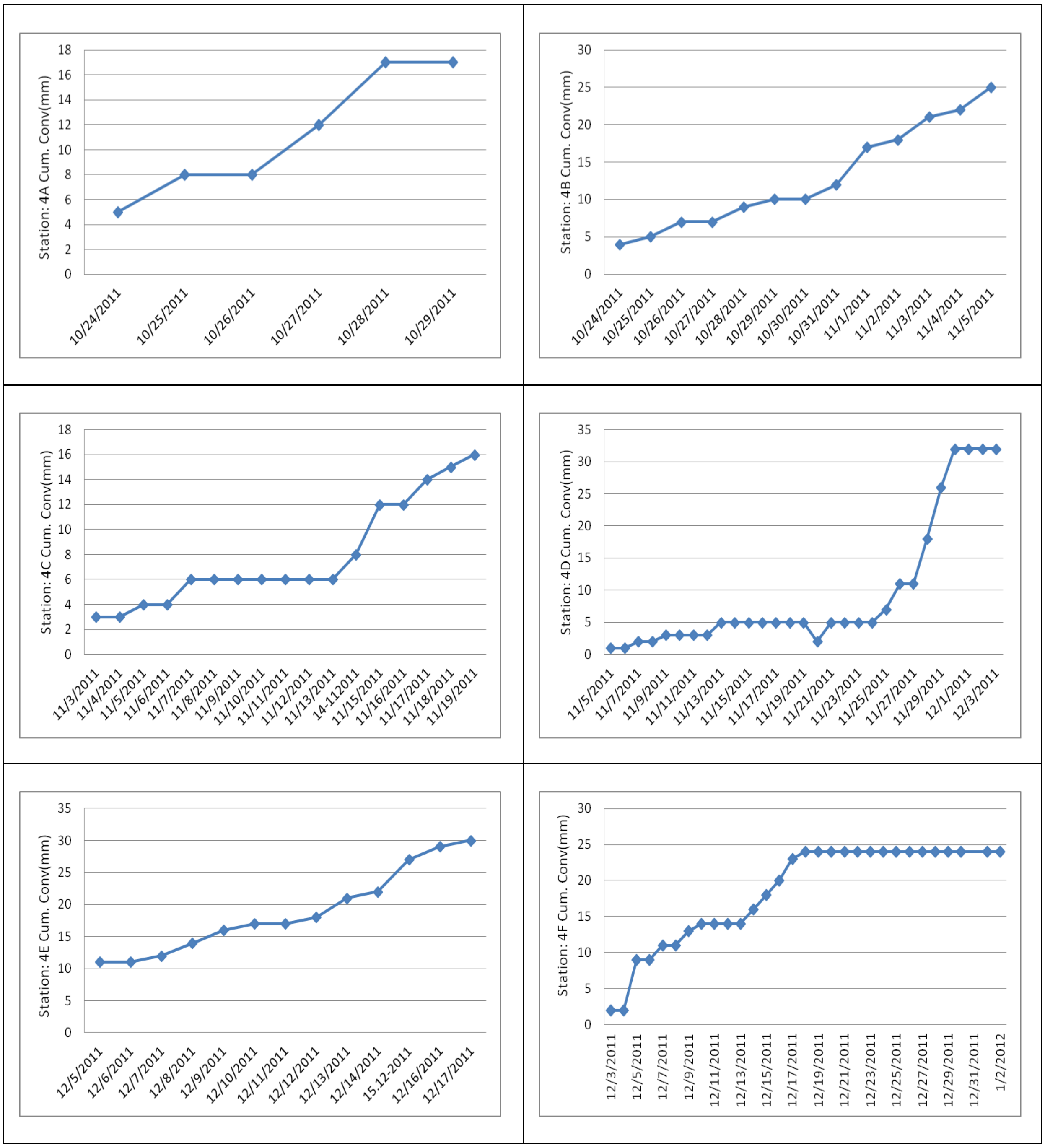


Figure 3.13: Convergence Observations in 58 LS

Maximum convergence observed at level 58LS was at station 4D installed on 5th November'2011 at a distance of about one pillar from the goaf edge. The maximum daily convergence recorded was 3 mm when the goaf edge was 15 m from the station. Total cumulative convergence recorded at this station was 32 mm. Maximum convergence was observed when station was near to the goaf edge.

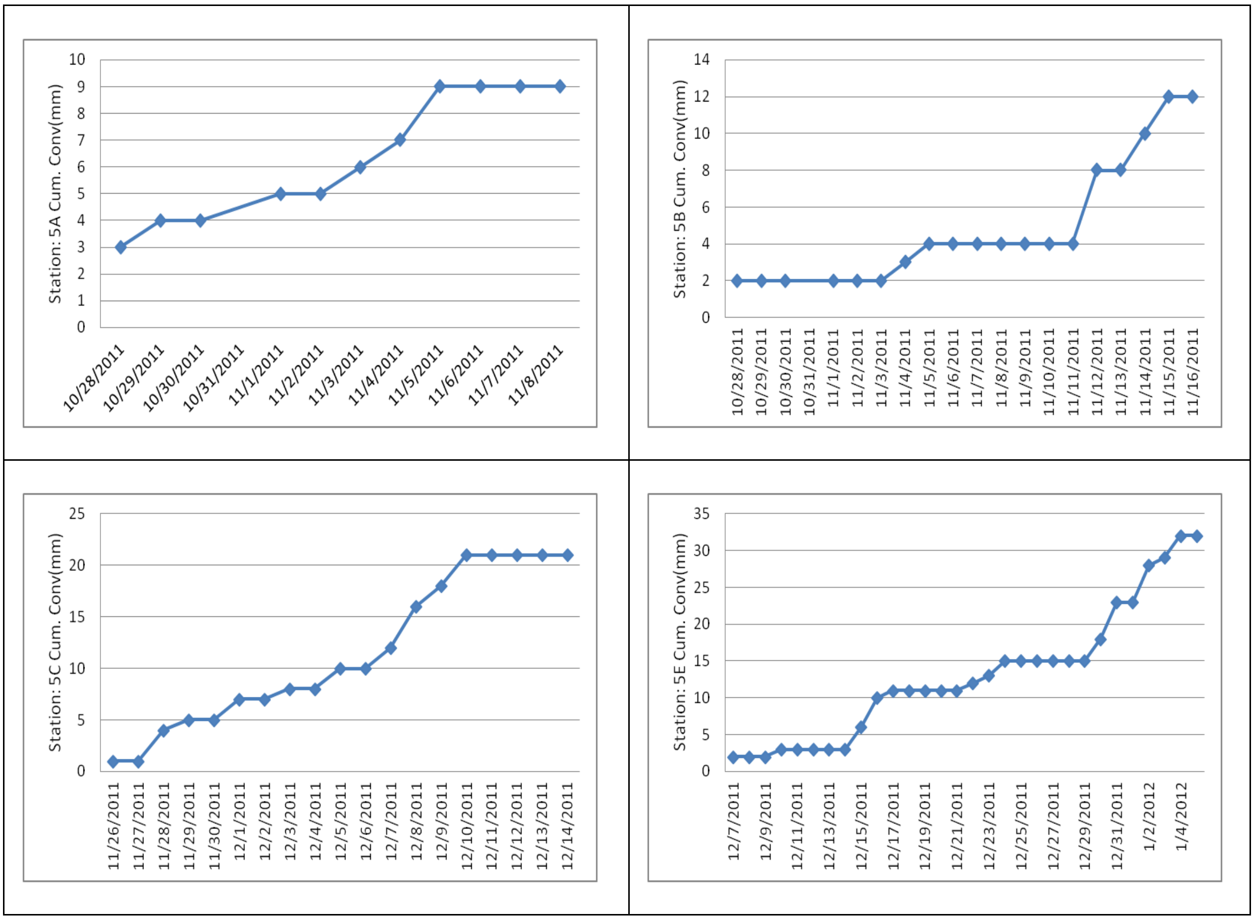
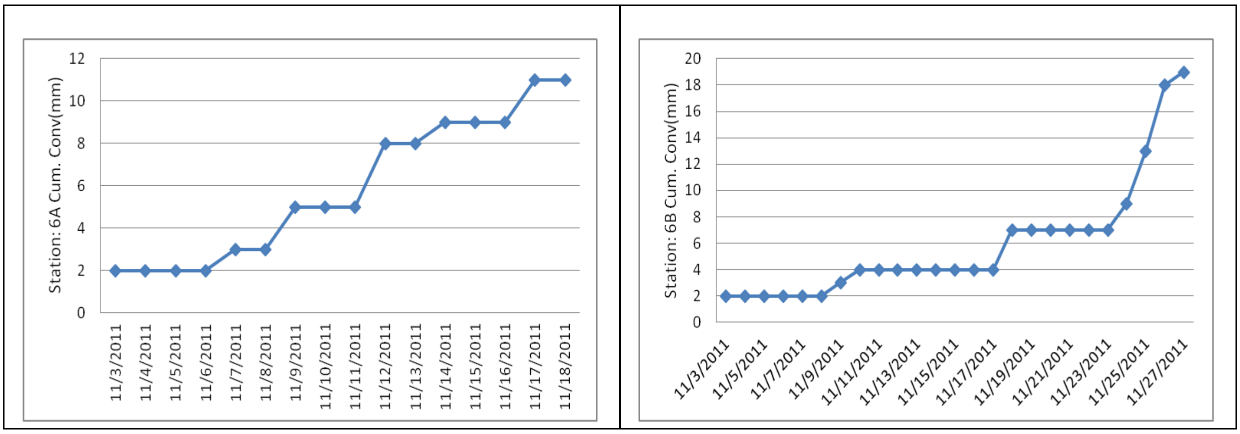


Figure 3.14: Convergence Observations in 57 ½ LS

Maximum convergence observed at level 57½LS was at station 5E installed on 7th December'2011 at a distance of about two pillar from the goaf edge. The maximum daily convergence recorded was 3 mm when the goaf edge was 15 m from the station. Total cumulative convergence recorded at this station was 31 mm. Maximum convergence was observed when station was near to the goaf edge.



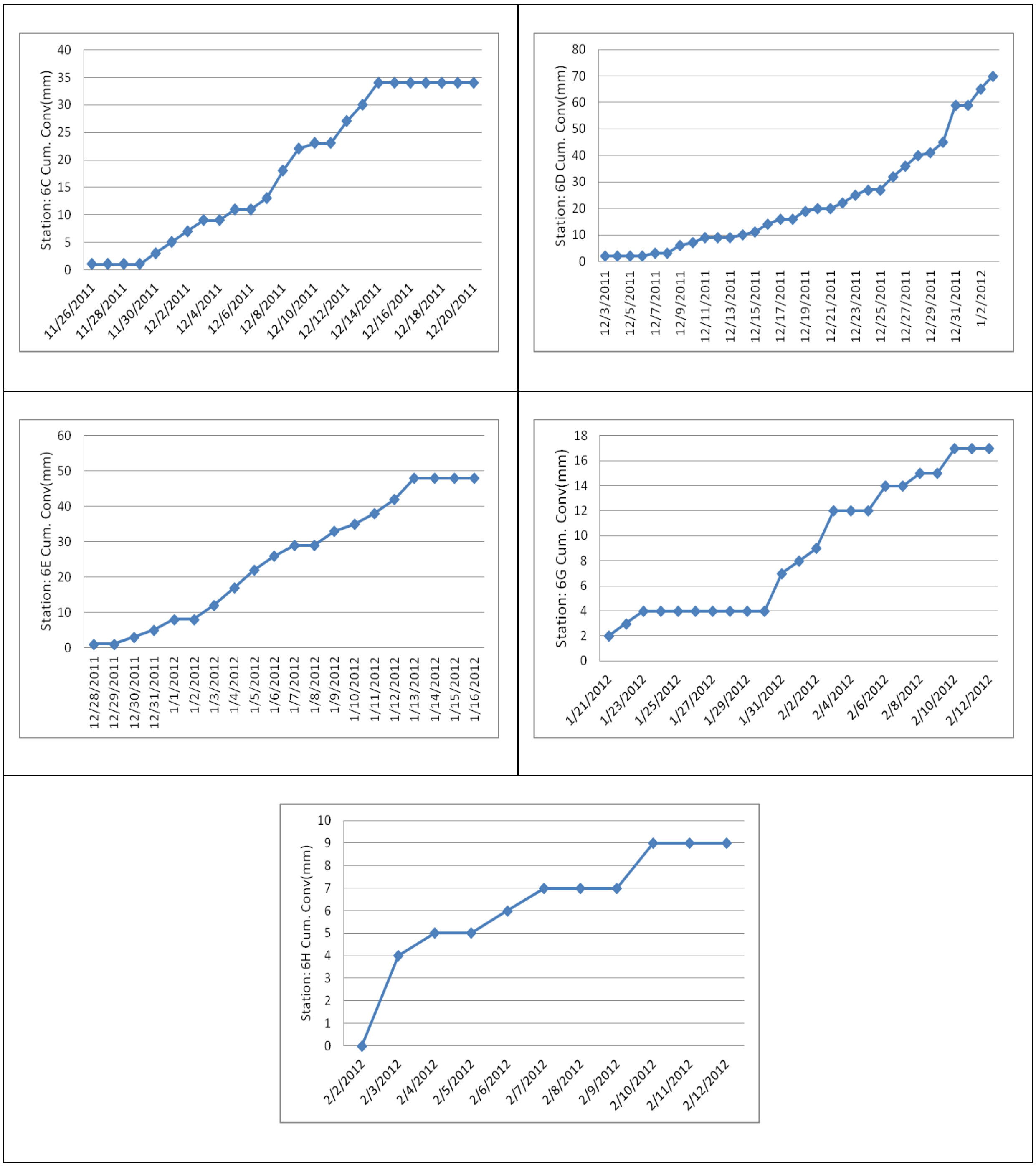


Figure 3.15: Convergence Observations in 57 LS

Maximum convergence observed at level 57LS was at station 6F installed on 13th December' 2011 at a distance of about two pillar from the goaf edge. The maximum daily convergence recorded was 3 mm when the goaf edge was 15 m from the station. Total cumulative convergence recorded at this station was 46 mm. Maximum convergence was observed when station was near to the goaf edge.

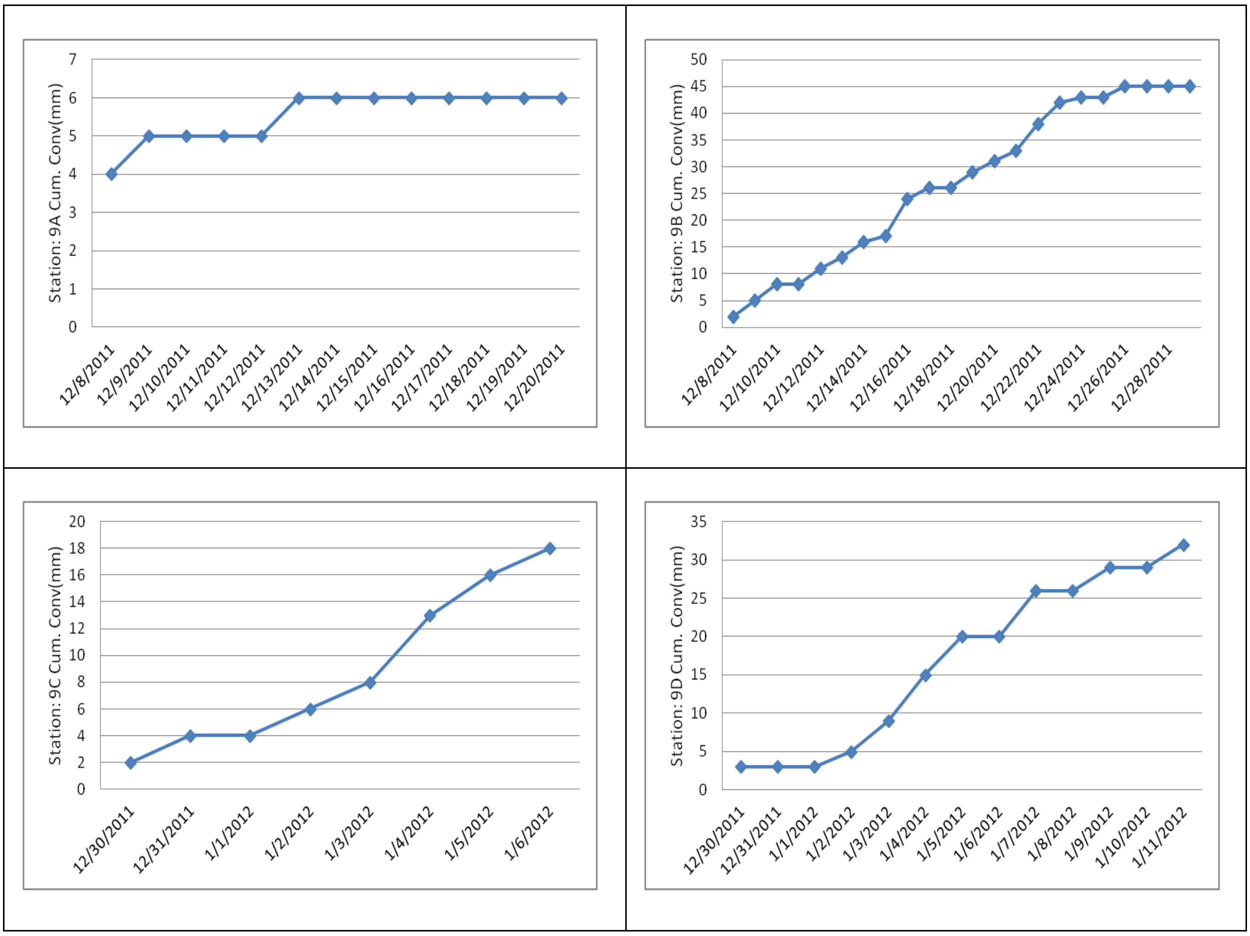
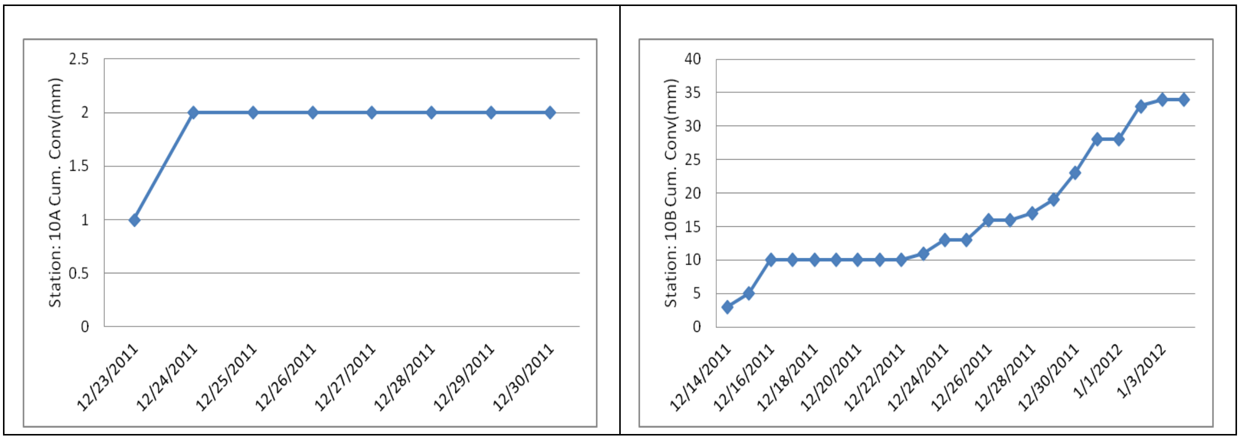


Figure 3.16: Convergence Observations in 55 ½ LS

Maximum convergence observed at level 55½LS was at station 9B installed on 13th December' 2011 at a distance of about one pillar from the goaf edge. The maximum daily convergence recorded was 3 mm when the goaf edge was 8 m from the station. Total cumulative convergence recorded at this station was 45 mm. Maximum convergence was observed when station was next to the goaf edge.



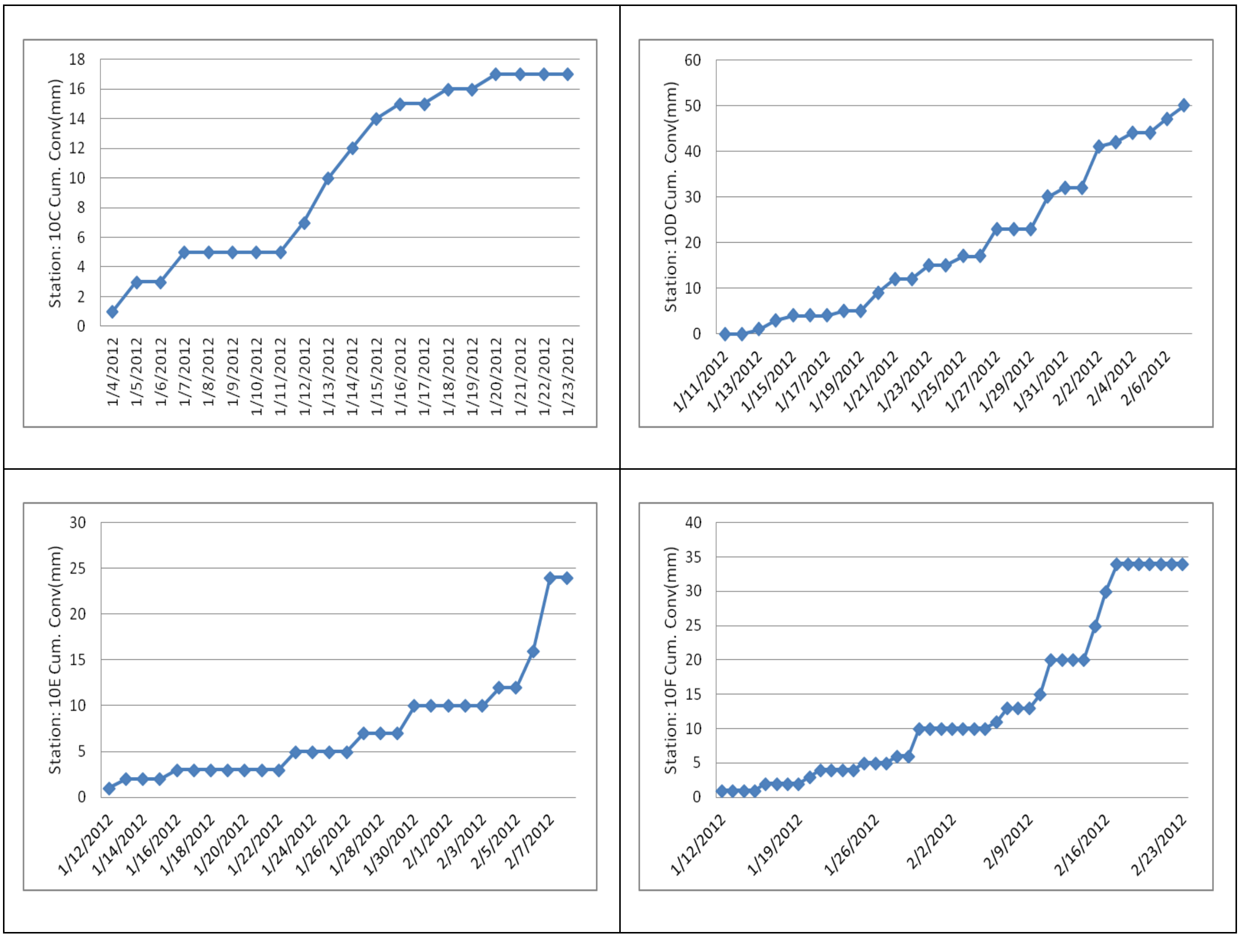
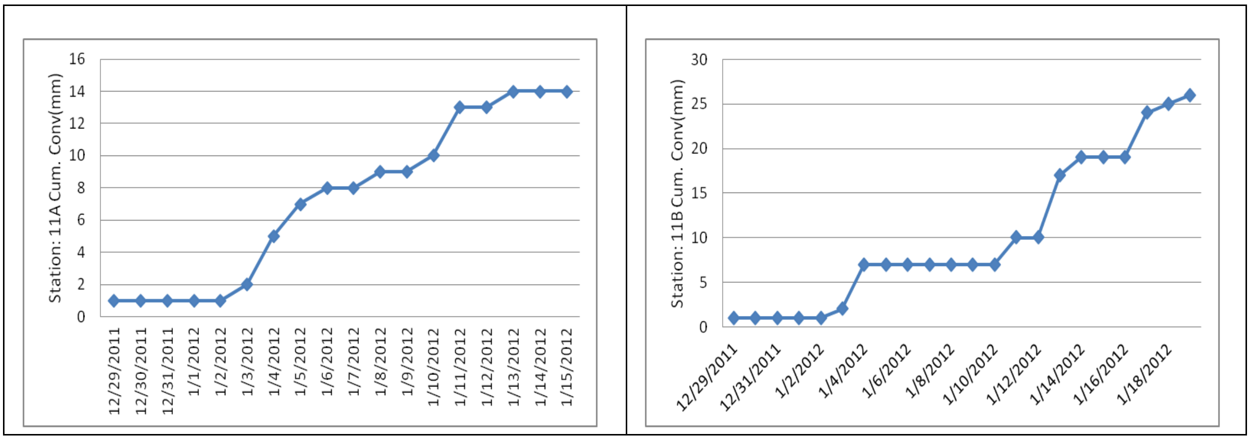


Figure 3.17: Convergence Observations in 55 LS

Maximum convergence observed at level 55LS was at station 10D installed on 11thJanuary' 2012 at a distance of about one pillar from the goaf edge. The maximum daily convergence recorded was 3 mm when the goaf edge was 6 m from the station. Total cumulative convergence recorded at this station was 48 mm. Maximum convergence was observed when station was next to the goaf edge.



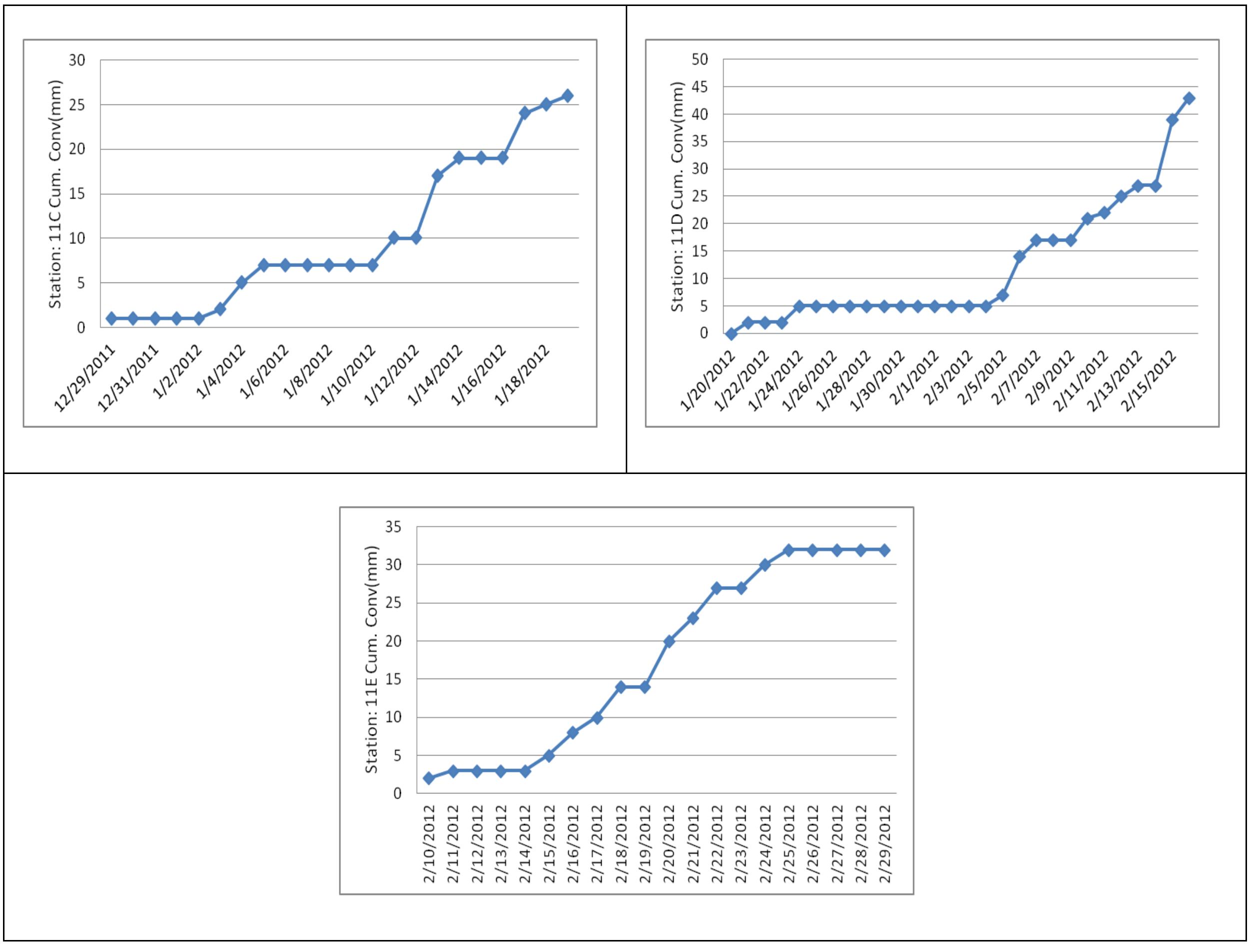


Figure 3.18: Convergence Observations in 54 ½ LS

Maximum convergence observed at level 54½LS was at station 11D installed on 20th January’ 2012 at a distance of about one pillar from the goaf edge. The maximum daily convergence recorded was 11 mm when the goaf edge was 4 m from the station. Total cumulative convergence recorded at this station was 43 mm. Maximum convergence was observed when station was next to the goaf edge.

CHAPTER 4

RESULTS AND ANALYSIS

RESULTS AND ANALYSIS

4.1 Convergence of the Galleries

Cumulative deformation of the FLAC simulation of numerical modeling for different stages is shown in Table 4.1. The model was simulated with roof support, roof and side support and without support to comprehend the strata condition.

Table 4.1: FLAC Simulation - Deformation Observation (mm)

Stage	Support	Gallery 1	Split 1	Gallery 2	Split 2	Gallery 3	Split 3	Gallery 4
Development of gallery	Without Support	8		8		8		8
	Roof bolting	5		5		5		5
	Roof and Side bolt	5		5		5		5
Development of splits	Without Support	8	8	8	8	8	8	8
	Roof bolting	5	5	5	5	5	5	5
	Roof and Side bolt	5	5	5	5	5	5	5
Extraction of stook 1	Without Support		15	10	10	10	10	10
	Roof bolting		11	8	8	8	8	8
	Roof and Side bolt		11	8	8	8	8	8
Extraction of stook 2	Without Support			20	15	15	15	15
	Roof bolting			14	11	11	11	11
	Roof and Side bolt			14	11	11	11	11
Extraction of stook 3	Without Support				30	20	20	20
	Roof bolting				22	14	14	14
	Roof and Side bolt				22	14	14	14
Extraction of stook 4	Without Support					44	30	30
	Roof bolting					34	22	22

	Roof and Side bolt					34	22	22
Extraction of stook 5	**Without Support**						60	44
	Roof bolting						45	34
	Roof and Side bolt						45	34
Extraction of stook 6	**Without Support**							80
	Roof bolting							58
	Roof and Side bolt							58

Numerical modeling results were compared with the field observation. Various stages of extraction in distance were correlated to the field working data in days. The X-axis corresponds to the days and Y-axis represents the cumulative convergence of the gallery.

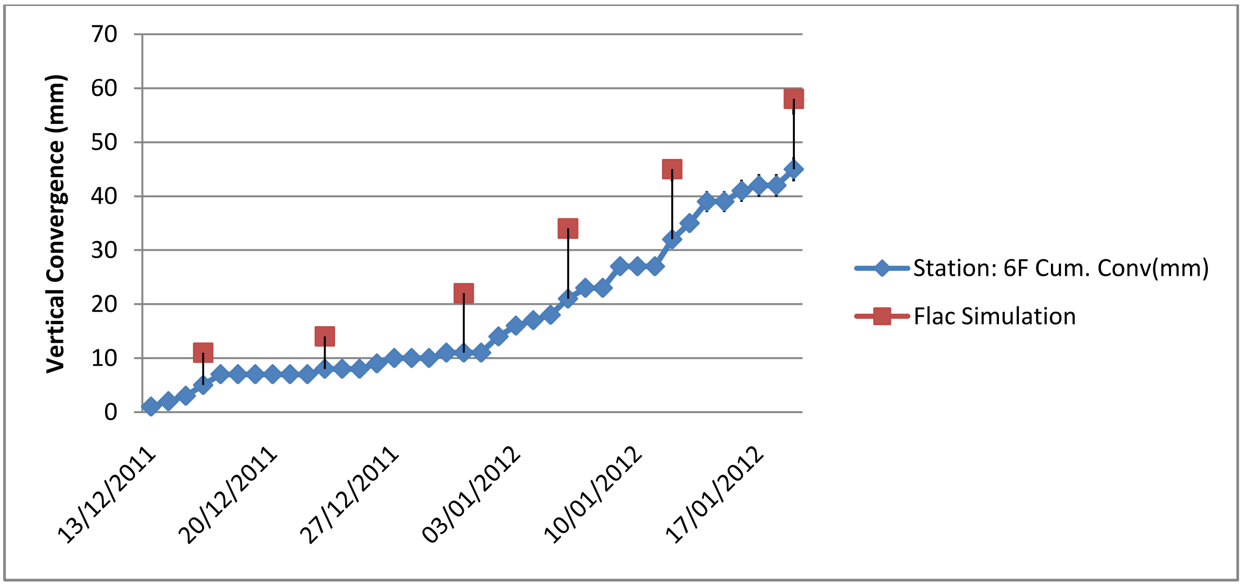


Figure 4.1: Comparison of Numerical Modeling and Field Observation

Comparison of numerical modeling cumulative deformation with field observation of cumulative convergence of galley adjoining goaf edge is presented in Figure 4.1 where Maximum deformation of 58mm was observed in the fourth gallery at stage of extraction of six stooks. Maximum convergence observed in the field was 48mm near to the goaf edge. The error of 17% convergence might be validated as the convergence reading in the field is not measured immediately after the opening of the gallery. Thus the deformation during the period between opening of gallery and installation of monitoring station may validate the larger deformation of the roof in numerical modeling.

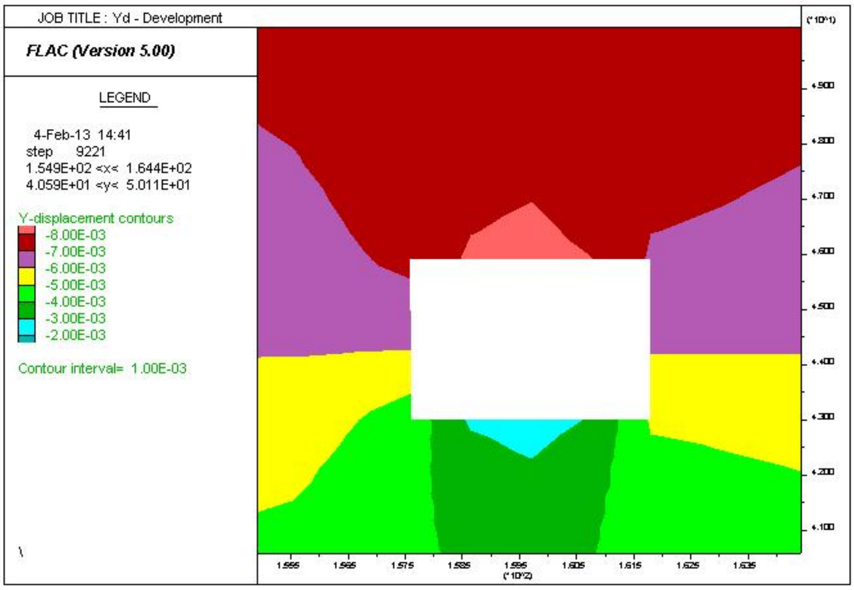


Figure 4.2: Roof Deformation in Gallery 1 of Unsupported Roof

Maximum convergence observed in unsupported gallery 1 at the stage of development of galleries is 8mm.

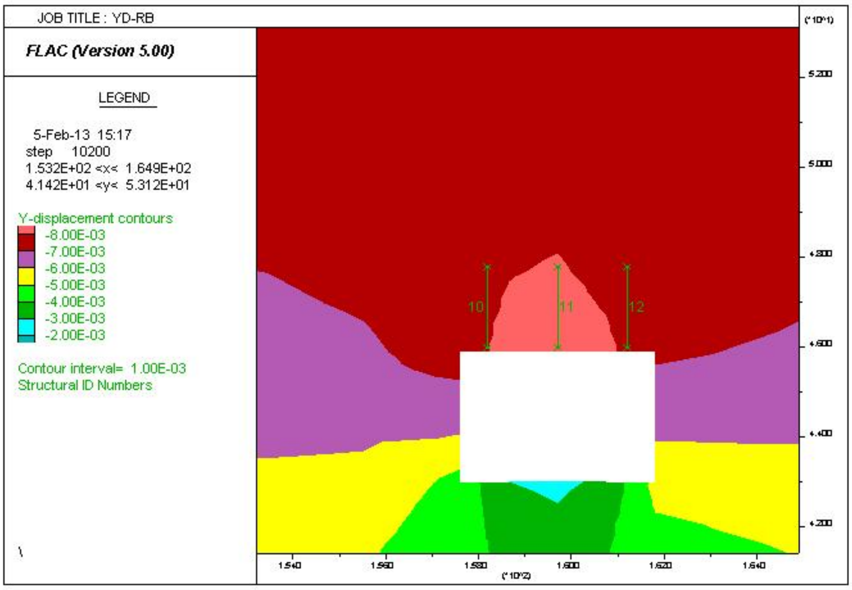


Figure 4.3: Roof Deformation in Gallery 2 Supported by Roof Bolt

Maximum deformation observed in the gallery 2 supported by roof bolt at the stage of extraction of 1 stook was 8mm.

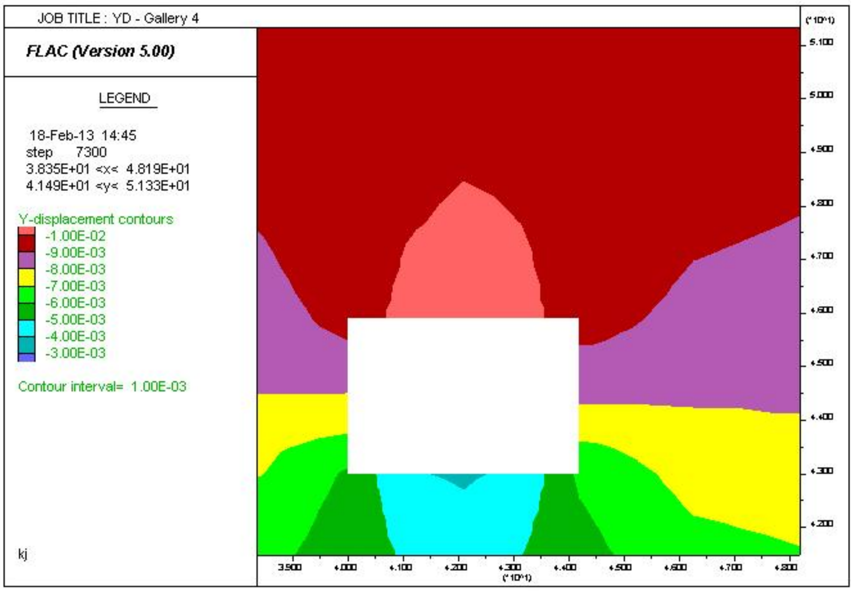


Figure 4.4: Roof Deformation in Gallery 2 Unsupported Roof

Maximum deformation observed in the gallery 2 unsupported roof at the stage of extraction of 1 stook was 10 mm.

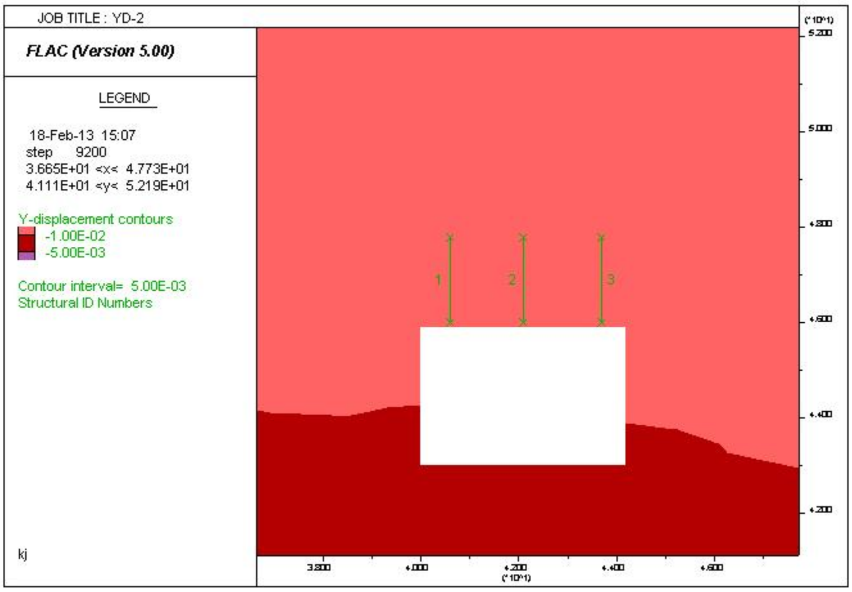


Figure 4.5: Roof Deformation in Split 2 Supported by Roof Bolt

Maximum deformation observed in the split 2 supported by roof bolt at the stage of extraction of 2 stook was 10 mm.

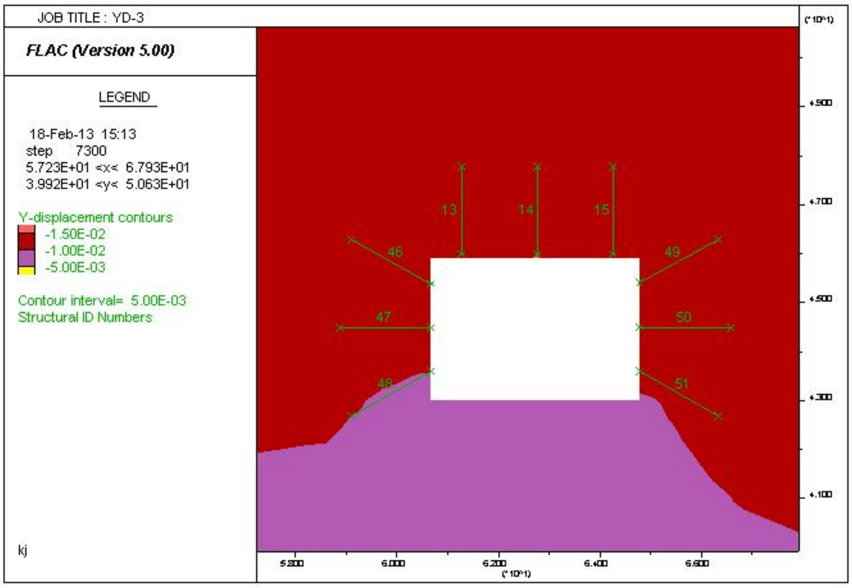


Figure 4.6: Roof Deformation in Gallery 2 Supported by Roof and Side Bolt

Maximum deformation observed in the gallery 2 supported by roof and side bolt at the stage of extraction of 2 stook was 15 mm.

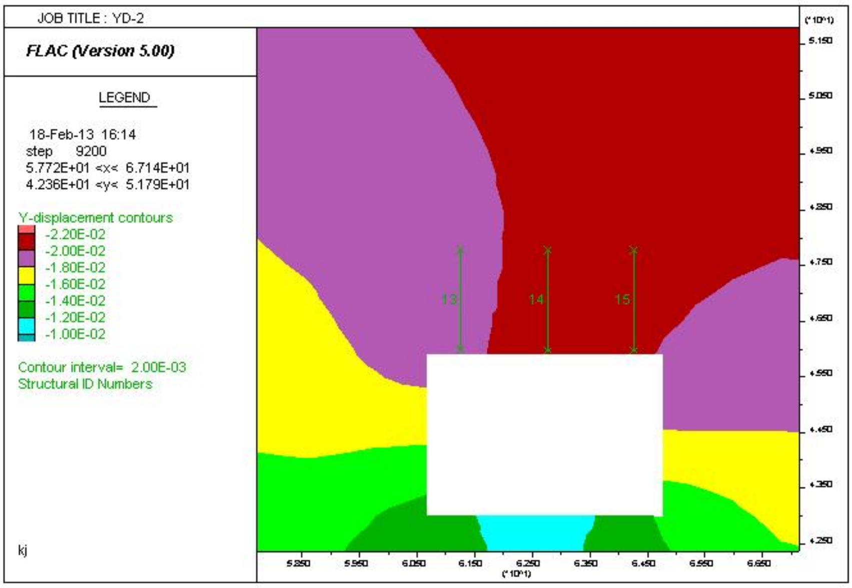


Figure 4.7: Roof Deformation in Split 3 Supported by Roof Bolt

Maximum deformation observed in the split 3 supported by roof bolt at the stage of extraction of 4 stook was 22 mm.

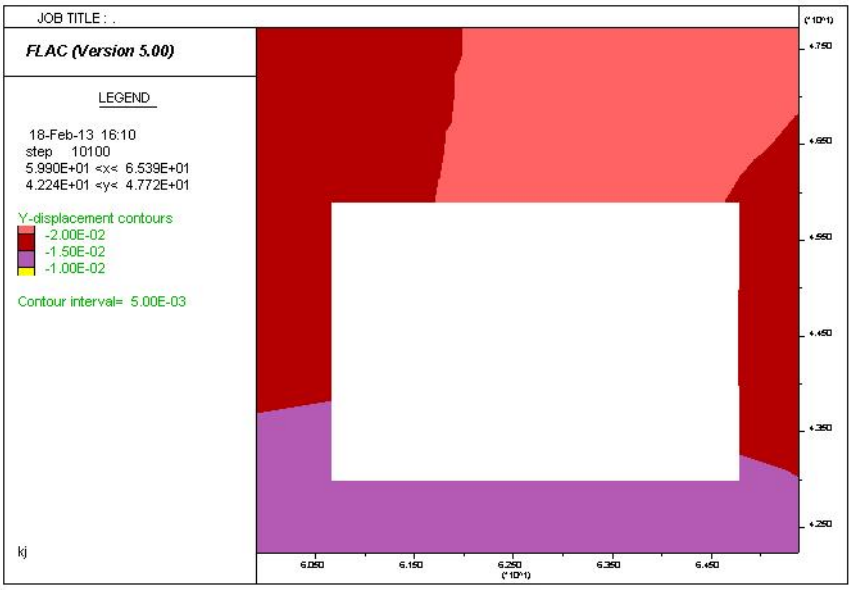


Figure 4.8: Roof Deformation in Gallery 3 Unsupported Roof

Maximum deformation observed in the gallery 3 unsupported roof at the stage of extraction of 3 stook was 20 mm.

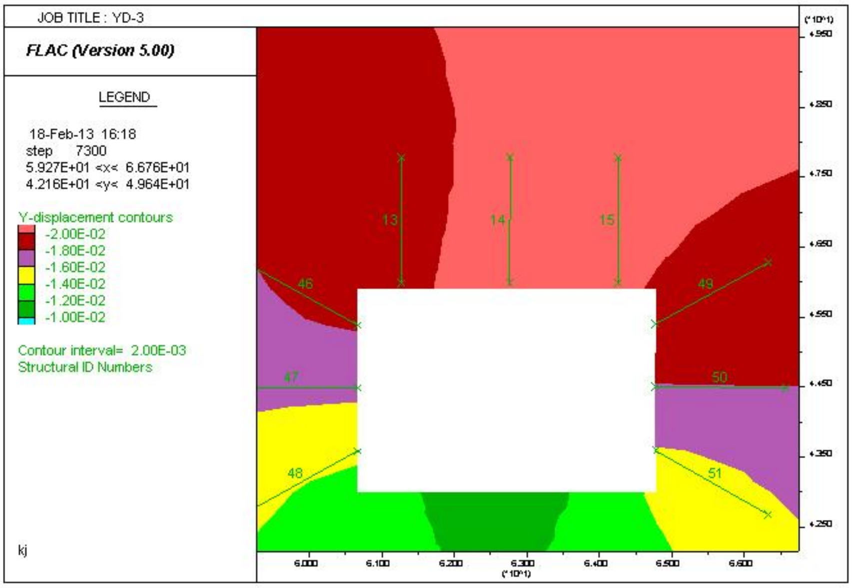


Figure 4.9: Roof Deformation in Gallery 4 Supported by Roof and Side Bolt

Maximum deformation observed in the gallery 4 supported by roof and side bolt at the stage of extraction of 4 stook was 20 mm.

The convergence trend of the galleries changes during the major fall due to release of abatement stresses of the un-collapsed roof in the goaf.

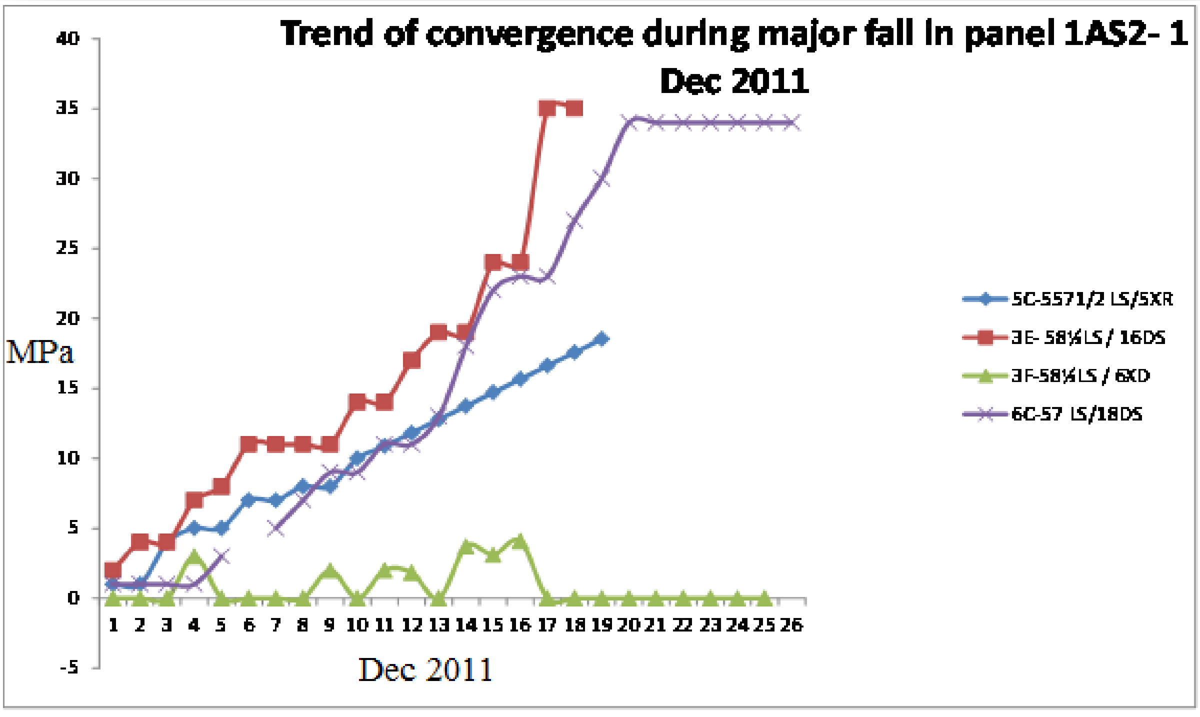


Figure 4.10: Trend of Convergence during Major Fall

The station 3F in Figure 4.10 shows little or no convergence because it is close to the goaf edge and it is relieved of abatement stresses due to major fall. Other stations 3E, 5C and 6C which are away from goaf edge shows increasing trend of convergence because of abatement loading of the uncollpased roof generated after extraction of pillars/stooks.

4.2 Stress Distribution over Pillars/Stooks

Cumulative stress over the pillars and stooks for FLAC simulation of numerical modeling for different stages is shown in Table 4.2. The model was simulated with roof support, roof and side support and without support to comprehend the stress distribution over the pillars and stooks.

Table 4.2: FLAC Simulation – Stress Observation (MPa)

<table>
<tr><th rowspan="2">Stage</th><th rowspan="2">Support</th><th colspan="2">Pillar 1</th><th colspan="2">Pillar 2</th><th colspan="2">Pillar 3</th></tr>
<tr><th>Stook 1</th><th>Stook 2</th><th>Stook 3</th><th>Stook 4</th><th>Stook 5</th><th>Stook 6</th></tr>
<tr><td rowspan="3">Development of gallery</td><td>Without Support</td><td colspan="2">5</td><td colspan="2">5</td><td colspan="2">5</td></tr>
<tr><td>Roof bolting</td><td colspan="2">5</td><td colspan="2">5</td><td colspan="2">5</td></tr>
<tr><td>Roof and Side bolt</td><td colspan="2">5</td><td colspan="2">5</td><td colspan="2">5</td></tr>
<tr><td>Development of splits</td><td>Without Support</td><td>5</td><td>5</td><td>5</td><td>5</td><td>5</td><td>5</td></tr>
</table>

	Roof bolting	5	5	5	5	5	5
	Roof and Side bolt	5	5	5	5	5	5
Extraction of stook 1	**Without Support**		8	6	5	5	5
	Roof bolting		8	6	5	5	5
	Roof and Side bolt		8	6	5	5	5
Extraction of stook 2	**Without Support**			8.5	7	5	5
	Roof bolting			8.5	7	5	5
	Roof and Side bolt			8.5	7	5	5
Extraction of stook 3	**Without Support**				9	7.5	6
	Roof bolting				9	7.5	6
	Roof and Side bolt				9	7.5	6
Extraction of stook 4	**Without Support**					9	8
	Roof bolting					9	8
	Roof and Side bolt					9	8
Extraction of stook 5	**Without Support**						9
	Roof bolting						9
	Roof and Side bolt						9

Maximum stress of 9 MPa is experienced by the stook present next to the fourth gallery after excavation of 5 stooks. The maximum over the pillar remains more or less same for supported and unsupported roof because the rock load remains constant. But the stress distribution profile changes showing more stress enforcement at the side of the pillars for supported roof and sides.

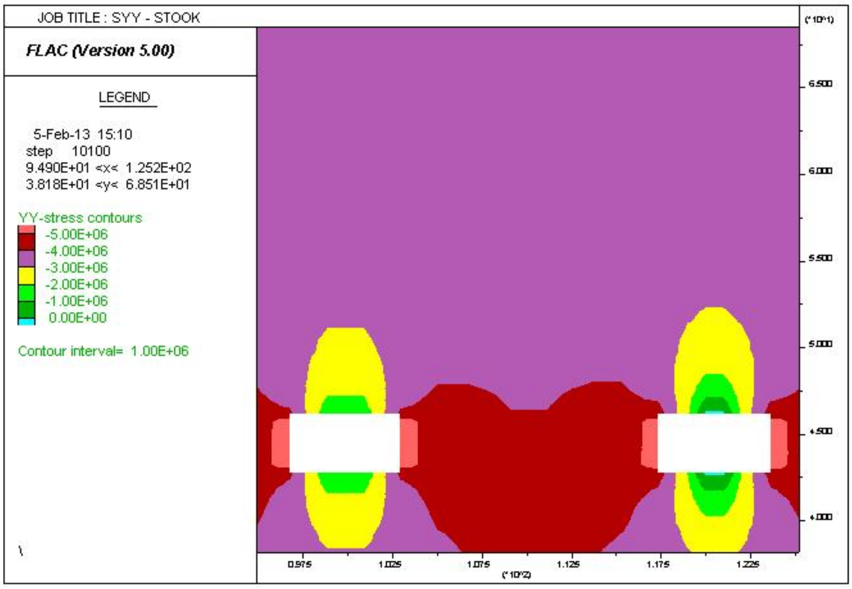


Figure 4.11: Stress Distribution over Pillar after Development of Splits

Maximum stress observed over the pillar at the development stage of splits was 5 MPa.

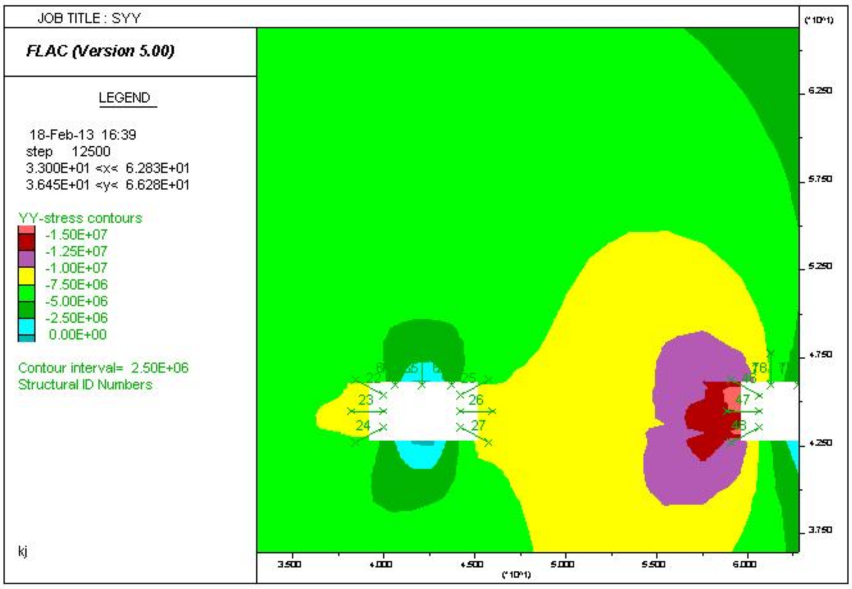


Figure 4.12: Stress Distribution over Stook 5

Maximum stress observed over the pillar at the stage of extraction of 5 stooks supported with roof and side bolts was 7.5 MPa.

Stress distribution over the fourth gallery is shown in the Figure 4.13. The X-axis represents the stress in MPa and Y-axis represents the goaf edge distance in meters.

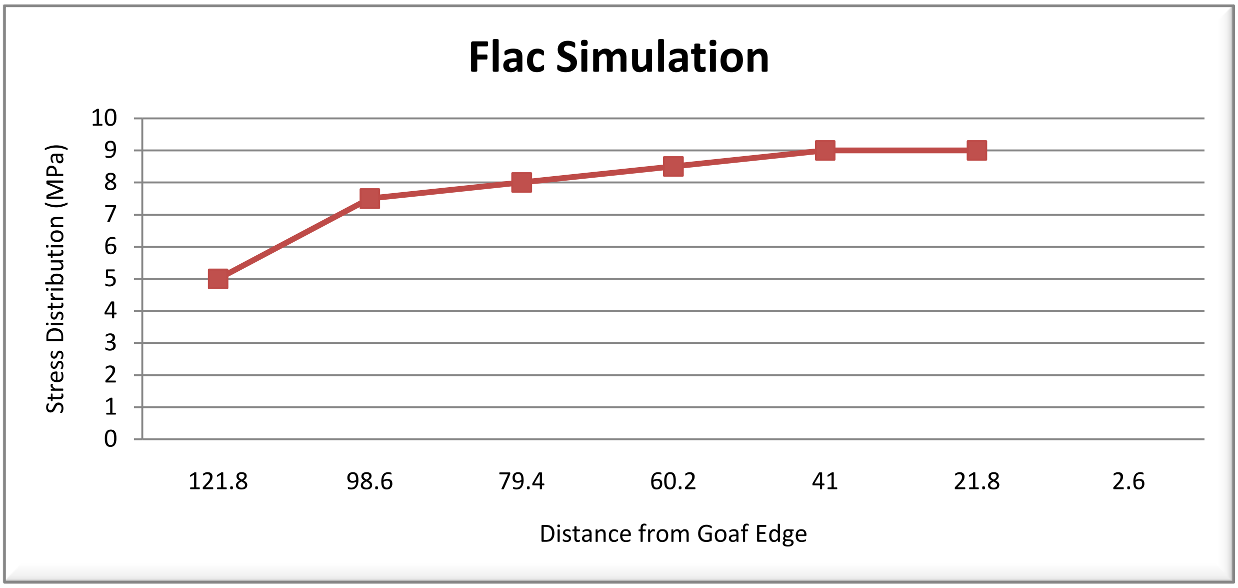


Figure 4.13: Stress Distribution over Pillar/Stook

The maximum stress distribution over the pillar/stook shows increasing trend because of load on the pillar/stook due to extraction of adjoining stooks. The maximum stress observed from modeling was 9 MPa.

CHAPTER 5

CONCLUSION

CONCLUSIONS

- Design of effective systematic support is essential for control of the strata and to provide safe working condition. The roof strata condition of 1AS2 panel of 1A seam RK-6 Incline was categorized as fair, as its RMR was 52. Design of systematic support by empirical approach yielded the following conclusions:

  Factor of safety of galleries, junctions, slices and goaf edges was calculated to be 2.05, 2.31, 1.75 and 1.39 respectively.
- The numeric modeling results were compared and validated with field monitoring data and following conclusions were drawn:

  Maximum cumulative convergence recorded in the field monitoring was 48mm in the 6F – 57LS, when the gallery was at the goaf edge. Results obtained from numerical modeling after implementation of the designed systematic support shows maximum convergence of 58mm in the gallery at the goaf edge.
- The model was almost validated with 17% approximation.
- Continuous increasing trend of the rate of convergence in the field is an indicator of impending major fall.
- Since the model is practically validated with field results, it can be used to predict the strata behavior of the working in advance.

5.1 Scope for Future Work

1. The approximation of validation can be reduced considerably through appropriate determination of the design parameters through laboratory studies.

2. 3- Dimensional models should be preferred over 2- Dimensional models to generate the geo-mining condition of the mine more effectively. 3-D model incorporates more complex geological features and provides flexibility of implementation of various supports in one gallery.

3. **3.** Effect of variation of different parameters on the strata stability can be studied. Results of models by varying geo-mining parameters can be validated with similar workings in various mines.

References

1. Singh, A.K., Sinha, A., Paul, A. and Saikia, K., Geotechnical investigation for support design in depillaring panels in Indian Coal mines. Journal of Scientific and Industrial Research. Volume 64, May 2005: pp. 358-363.
2. Cambulat, I., Evaluation and design of Optimum Support System in South African Collieries using Probabilistic design approach. University of Pretoria, 2008.
3. Palei, S.K. and Das, S.K., Sensitivity analysis of Support Safety Factor for predicting the effect of contributing parameters on roof falls in underground coal mines. International Journal of Coal Geology, 75 (2008): pp. 241-247.
4. Palei, S.K. and Das, S.K., Logistic regression model for prediction of roof fall risk in Bord and Pillar workings in Coal mines. Safety science, 47 (2009): pp. 88-96.
5. Cambulat, I., Roadway roof support designing in critical areas at Anglo American Metallurgical Coal’s underground in Aziz. 10th Underground coal operators Conference (2010): pp. 50-72.
6. Singh, A.K., Singh R, Maiti, J, Kumar, R and Mandal, P.K., Assessment of mining induced stresses developed over coal pillars during depillaring, International Journal of Rock Mechanics And Mining Sciences, 48 (2011): pp. 805-818.
7. Singh, A.K., Singh, R, Maiti, J, Kumar, R and Sinha, A., Coal pillar extraction at deep cover; with special reference to Indian coalfields. International Journal Of Coal Geology, Volume 86, Issue 2-3, May 2011: pp. 276-288.
8. Maiti, J, and Khanzode, V.V., development of relative risk model for roof and side fall fatal accidents in underground coal mines in India. Safety Science, 47 (2011): pp. 1068-1076
9. Khushwaha, A, Singh, S.K., Tewari, S. and Sinha, A., Empirical approach for designing of support system in mechanized coal pillar mining. International Journal of Rock Mechanics and Mining Sciences, 47 (2011): pp. 1063-1078.
10. Singh, R, Singh, A.K., Maiti, J, Mandal, P.K. and Kumar, R, An observational approach for assessment of dynamic loading during underground coal pillar extraction. International Journal of Rock Mechanics and Mining Sciences, 48 (2011): pp.794-804.
11. Jayanthu, S., Lakshminarayana, V., Singh, T.N. and Singh, D.P., Organization of strata monitoring cell – A vital requirement for all underground coal mines. The Indian Mining and Engineering Journal. Volume 51, July 2012: pp. 23-31.

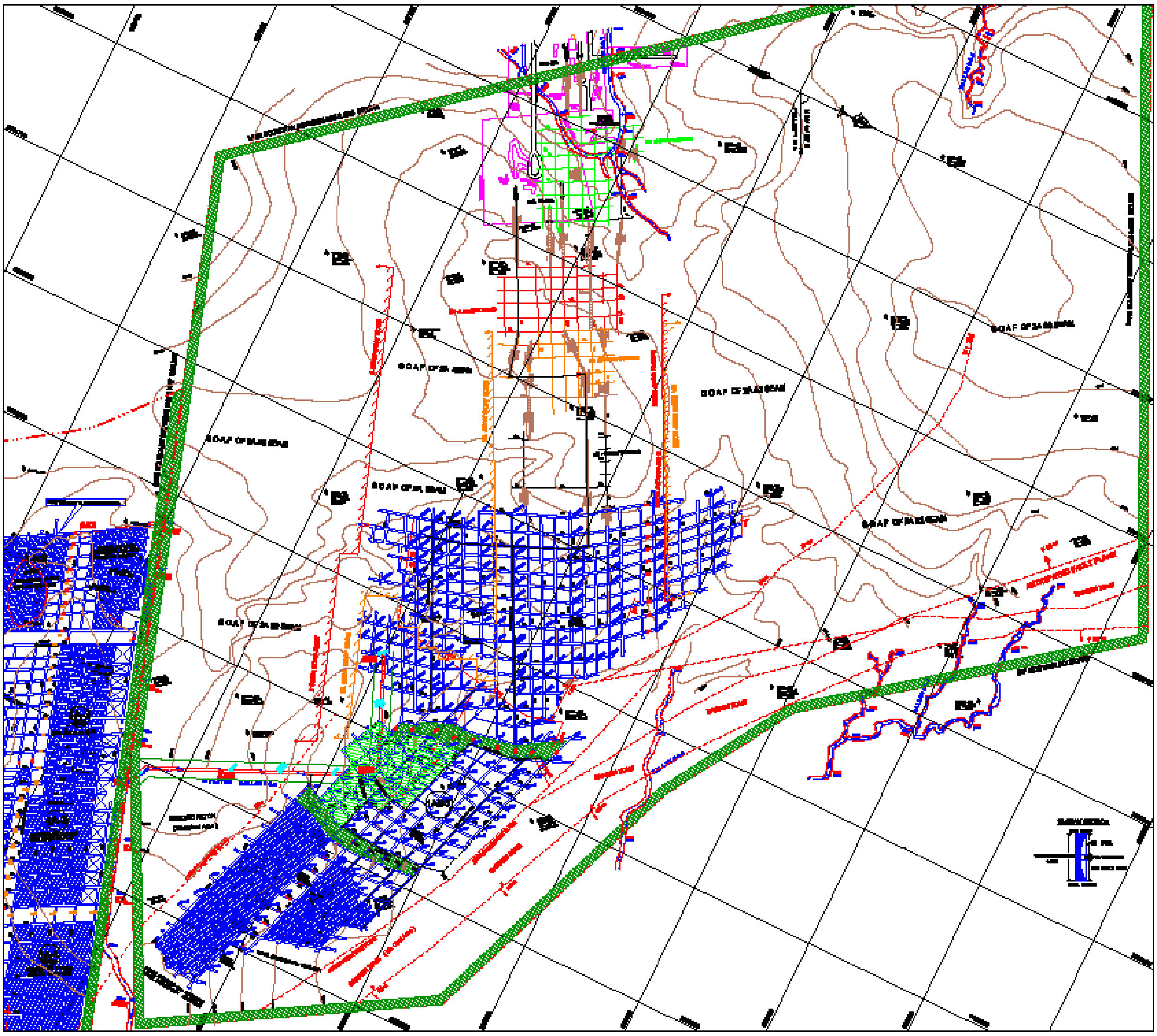
12. Maiti, J., Mandal, P.K., Singh, R. and Kumar, B.M., Impact of stress redistribution on stability of workings during depillaring. IE (1) Journal – MN: pp. 10-22.
13. Fast Lagrangian Analysis of Continua Manual, Itasca Consulting Group Inc., Second Edition (*FLAC* Version 5.0) April 2005.
14. DGMS Annual Report, 2007: pp. ii.
15. Bieniawski, Z. T., 'Rock Mass Classification in Rock Engineering'. Proceedings of the Symposium on Exploration for Rock Engineering, Johannesburg, South Africa, 1976: pp. 97.

ANNEXURE 1

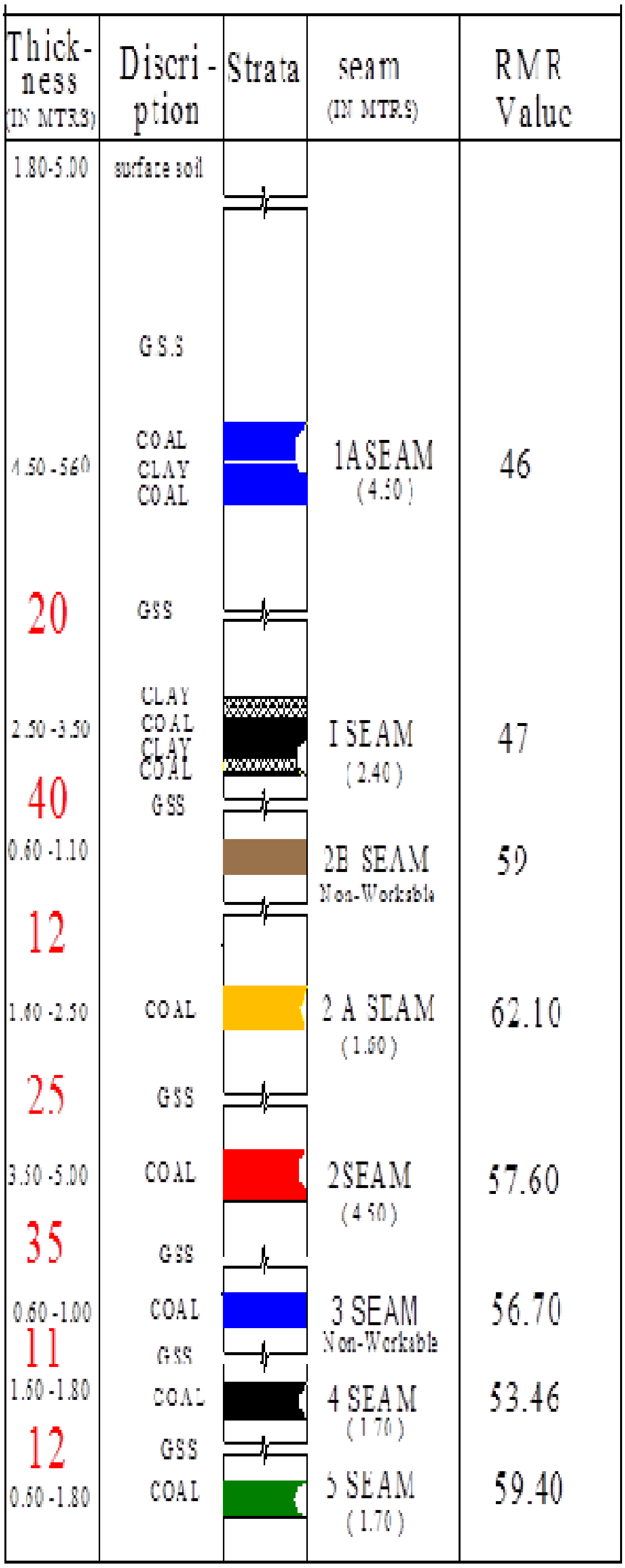
Mine Details

Details of the 1AS2 Panel of RK-6 Incline

RavindraKhani No.6 Incline is situated 11 Kms from Mancherial Railway station. It falls in Indaram and North Godavari Mining lease of S.C.C.L. The lease hold area of the mine is 306 Hec in forest land. The mine was started in the year 1975. Presently 5 coal seams are working with 4 Hand section drills and 8 SDL's. RK6 Incline having 8 coal seams. The average gradient of the seam is 1 in 4 the details are as follows.



Mine Plan of RK-6 Incline



General Bore Hole Section of RK-6 Inc

ANNEXURE II

Model Code

```
TITLE

DESIGN OF SUPPORT SYSTEM IN DEVELOPEMENT WORKINGS IN RK6 INCLINE

*PROGRAM DEVELOPED BY VIKRANT DEV SINGH

* Seam thickness=5.5m, Pillar size=30m, Depth=160m

* Gallery size=4.2m X 3m

GR 87 44

M M

*

*FLOOR OF THE MODEL

gen 0,0 0,40 40,40 40,0                         R .8 .8 I 1 10 J 1 10

gen 40,0 40,40 44.2,40 44.2,0                   R 1 .8 I 10 14 J 1 10

gen 44.2,0 44.2,40 79.2,40 79.2,0               R 1 .8 I 14 31 J 1 10

gen 79.2,0 79.2,40 83.4,40 83.4,0               R 1 .8 I 31 35 J 1 10

gen 83.4,0 83.4,40 118.4,40 118.4,0             R 1 .8 I 35 52 J 1 10

gen 118.4,0 118.4,40 122.6,40 122.6,0           R 1 .8 I 52 56 J 1 10

gen 122.6,0 122.6,40 157.6,40 157.6,0           R 1 .8 I 56 73 J 1 10

gen 157.6,0 157.6,40 161.8,40 161.8,0           R 1 .8 I 73 77 J 1 10

gen 161.8,0 161.8,40 201.8,40 201.8,0           R 0.8 .8 I 77 88 J 1 10

*

*Coal seam -5.5m

gen 0,40 0,45.5 40,45.5 40,40                   R .8 1 I 1 10 J 10 21
```

```
gen 40,40 40,45.5 44.2,45.5 44.2,40             R 1 1 I 10 14 J 10 21

gen 44.2,40 44.2,45.5 79.2,45.5 79.2,40         R 1 1 I 14 31 J 10 21

gen 79.2,40 79.2,45.5 83.4,45.5 83.4,40         R 1 1 I 31 35 J 10 21

gen 83.4,40 83.4,45.5 118.4,45.5 118.4,40       R 1 1 I 35 52 J 10 21

gen 118.4,40 118.4,45.5 122.6,45.5 122.6,40R 1 1 I 52 56 J 10 21

gen 122.6,40 122.6,45.5 157.6,45.5 157.6,40R 1 1 I 56 73 J 10 21

gen 157.6,40 157.6,45.5 161.8,45.5 161.8,40R 1 1 I 73 77 J 10 21

gen 161.8,40 161.8,45.5 201.8,45.5 201.8,40R 0.8 1 I 77 88 J 10 21

*

* Sandstone roof-

gen 0,45.5 0,205.5 40,205.5 40,45.5             R .8 1.2 I 1 10 J 21 45

gen 40,45.5 40,205.5 44.2,205.5 44.2,45.5       R 1 1.2 I 10 14 J 21 45

gen 44.2,45.5 44.2,205.5 79.2,205.5 79.2,45.5R 1 1.2 I 14 31 J 21 45

gen 79.2,45.5 79.2,205.5 83.4,205.5 83.4,45.5R 1 1.2 I 31 35 J 21 45

gen 83.4,45.5 83.4,205.5 118.4,205.5 118.4,45.5 R 1 1.2 I 35 52 J 21 45

gen 118.4,45.5 118.4,205.5 122.6,205.5 122.6,45.5  R 1 1.2 I 52 56 J 21 45

gen 122.6,45.5 122.6,205.5 157.6,205.5 157.6,45.5  R 1 1.2 I 56 73 J 21 45

gen 157.6,45.5 157.6,205.5 161.8,205.5 161.8,45.5  R 1 1.2 I 73 77 J 21 45

gen 161.8,45.5 161.8,205.5 201.8,205.5 201.8,45.5  R 0.8 1.2 I 77 88 J 21 45

PROP S=42E9 B=6.67E9 D=2100 T=9E6 C= 6.75E6 FRIC=45 I 1 87 J 1 9

PROP S=4E9 B=6.67E9 D=2100 T=9E6 C= 6.75E6 FRIC=45 I 1 87 J 21 44

PROP S=2.2E9 B=3.67E9 D=1480 T=1.86E6 C=1.85E6 FRIC=30 I 1 87 J 10 20

PROP S=1.4E9 B=2E9 D=1650 T=6000 C=5000 FRIC=17 I 1 87 J 17
```

```
SET GRA 9.81

set large

FIX X I 1

FIX X J 1

FIX X I 88

FIX Y J 1

INI SYY -3.38E6 VAR 0 3.38E6

INI SXX -1.45E6 VAR 1.45 1.45E6

HIS NSTEP 10

*Development galleries 4m x 3m

HIS UNBAL I 1 J 1

********************OPENING OF GALLERY 1***********

MOD NULL I 10 13 J 16 21

********************OPENING OF GALLERY 2***********

MOD NULL I 31 34 J 16 21

********************OPENING OF GALLERY 3***********

MOD NULL i 52 55 J 16 21

********************OPENING OF GALLERY 4***********

MOD NULL i 73 76 J 16 21

********************OPENING OF SPLIT 1***********

MOD NULL I 22 23 J 16 21

********************OPENING OF SPLIT 2***********

MOD NULL I 42 43 J 16 21
```

*******************OPENING OF SPLIT 3***********

MOD NULL i 63 64 J 16 21

*Excavation of stook 1

MOD NULL i 66 76 j 16 21

S=100

SAVE rk6s1.sav

***********Gallery 1 ROOF BOLTS

STRUCT CABL BEG 40.6 46. END 40.6 47.8. PROP 1

STRUCT CABL BEG 42.1 46. END 42.1 47.8. PROP 1

STRUCT CABL BEG 43.7 46. END 43.7 47.8. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********Gallery 2 rb

STRUCT CABL BEG 79.8 46. END 79.8 47.8. PROP 1

STRUCT CABL BEG 81.3 46. END 81.3 47.8. PROP 1

STRUCT CABL BEG 82.8 46. END 82.8 47.8. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********Gallery 3 rb

STRUCT CABL BEG 119 46. END 119 47.8. PROP 1

STRUCT CABL BEG 120.5 46. END 120.5 47.8. PROP 1

STRUCT CABL BEG 122 46. END 122 47.8. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********Gallery 4 rb

STRUCT CABL BEG 158.2 46. END 158.2 47.8. PROP 1

STRUCT CABL BEG 159.7 46. END 159.7 47.8. PROP 1

STRUCT CABL BEG 161.2 46. END 161.2 47.8. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********SPLIT 1 ROOF BOLTS

STRUCT CABL BEG 61.27 46. END 61.27 47.8. PROP 1

STRUCT CABL BEG 62.77 46. END 62.77 47.8. PROP 1

STRUCT CABL BEG 64.27 46. END 64.27 47.8. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********SPLIT 2 rb

STRUCT CABL BEG 98.41 46. END 98.41 47.8. PROP 1

STRUCT CABL BEG 99.91 46. END 99.91 47.8. PROP 1

STRUCT CABL BEG 101.41 46. END 101.41 47.8. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********SPLIT 3 rb

STRUCT CABL BEG 137.61 46. END 137.61 47.8. PROP 1

STRUCT CABL BEG 139.11 46. END 139.11 47.8. PROP 1

STRUCT CABL BEG 140.61 46. END 140.61 47.8. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

S=100

SAVE RK6S10.SAV

***********Gallery 1 sb

STRUCT CABL BEG 38.44 46.3. END 40 45.4. PROP 1

STRUCT CABL BEG 38.2 44.5. END 40 44.5. PROP 1

STRUCT CABL BEG 38.44 42.7. END 40 43.6. PROP 1

STRUCT CABL BEG 44.2 45.4. END 45.76 46.3. PROP 1

STRUCT CABL BEG 44.2 44.5. END 46 44.5. PROP 1

STRUCT CABL BEG 44.2 43.6. END 45.76 42.7. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********Gallery 2 sb

STRUCT CABL BEG 76.64 46.3. END 79.2 45.4. PROP 1

STRUCT CABL BEG 77.4 43.5. END 79.2 44.5. PROP 1

STRUCT CABL BEG 77.64 42.7. END 79.2 43.6. PROP 1

STRUCT CABL BEG 83.4 45.4. END 84.96 46.3. PROP 1

STRUCT CABL BEG 83.4 44.5. END 85.2 44.5. PROP 1

STRUCT CABL BEG 83.4 43.6. END 84.96 42.7. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********Gallery 3 sb

STRUCT CABL BEG 116.84 46.3. END 118.4 45.4. PROP 1

STRUCT CABL BEG 116.6 44.5. END 118.4 44.5. PROP 1

STRUCT CABL BEG 116.84 42.7. END 118.4 43.6. PROP 1

STRUCT CABL BEG 122.6 45.4. END 124.16 46.3. PROP 1

STRUCT CABL BEG 122.6 44.5. END 124.4 44.5. PROP 1

STRUCT CABL BEG 122.6 43.6. END 124.16 42.7. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********Gallery 4 sb

STRUCT CABL BEG 156.04 46.3. END 157.6 45.4. PROP 1

STRUCT CABL BEG 155.8 44.5. END 157.6 44.5. PROP 1

STRUCT CABL BEG 156.04 42.7. END 157.6 43.6. PROP 1

STRUCT CABL BEG 161.8 45.4. END 163.36 46.3. PROP 1

STRUCT CABL BEG 161.8 44.5. END 163.6 44.5. PROP 1

STRUCT CABL BEG 161.8 43.6. END 163.36 42.7. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********SPLIT 1 sb

STRUCT CABL BEG 59.11 46.3. END 60.67 45.4. PROP 1

STRUCT CABL BEG 58.87 44.5. END 60.67 44.5. PROP 1

STRUCT CABL BEG 59.11 42.7. END 60.67 43.6. PROP 1

STRUCT CABL BEG 64.78 45.4. END 66.34 46.3. PROP 1

STRUCT CABL BEG 64.78 44.5. END 66.58 44.5. PROP 1

STRUCT CABL BEG 64.78 43.6. END 66.34 42.7. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********SPLIT 2 sb

STRUCT CABL BEG 96.25 46.3. END 97.81 45.4. PROP 1

STRUCT CABL BEG 96.01 44.5. END 97.81 44.5. PROP 1

STRUCT CABL BEG 96.25 42.7. END 97.81 43.6. PROP 1

STRUCT CABL BEG 101.93 45.4. END 103.49 46.3. PROP 1

STRUCT CABL BEG 101.93 44.5. END 103.73 44.5. PROP 1

STRUCT CABL BEG 101.93 43.6. END 103.49 42.7. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

***********SPLIT 3 sb

```
STRUCT CABL BEG 135.44 46.3. END 137 45.4. PROP 1

STRUCT CABL BEG 135.2 44.5. END 137 44.5. PROP 1

STRUCT CABL BEG 135.44 42.7. END 137 43.6. PROP 1

STRUCT CABL BEG 141.12 45.4. END 142.68 46.3. PROP 1

STRUCT CABL BEG 141.12 44.5. END 142.82 44.5. PROP 1

STRUCT CABL BEG 141.12 43.6. END 142.68 42.7. PROP 1

STRUCT PROP 1 YI 1E6 KB 1E9 SB 1E7 E 200E9 A 3.14E-4

S=100

SAVE RK6S11.SAV

RET
```